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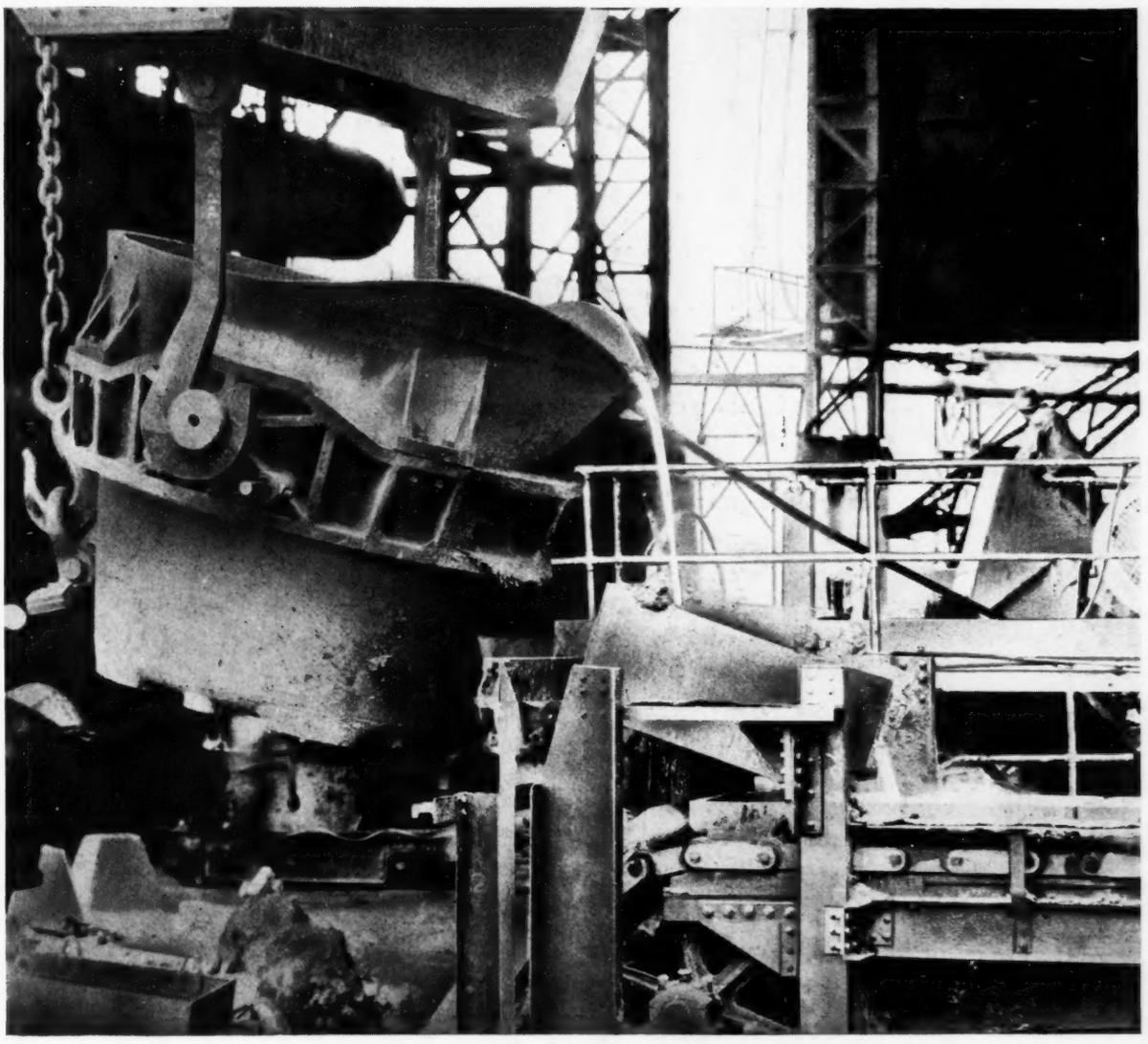

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October, 1932

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Engineering and Mining Journal, October, 1932. Vol. 133, No. 10. Published monthly by the McGraw-Hill Publishing Company, Inc., 330 West 42d St., New York City, N. Y., U. S. A. Subscription rates, United States, $3 per annum; Canada and other countries in the Americas outside the United States, $4 per annum; elsewhere $5 or 25 shillings per annum. Price per copy, 50 cents or 2s. 6d. Entered as second-class matter June 20, 1879, at the Post Office, New York, N. Y., under the act of March 3, 1879. Printed in U. S. A.

Engineering and Mining Journal

with which is consolidated

Engineering & Mining World

October, 1932 Vol. 133—No. 10

A. W. ALLEN, *Editor*

Metals and Minerals In House Construction

WHATEVER the course of future events in industry, one may reasonably predict a trend toward better housing, whether provided by an individual for himself, by a builder for general sale, or by a company for its employees. The public is rapidly learning what good housing means. Most dwellings are hopelessly deficient in respect to heat insulation. Improvement in housing aside from this particular might also be forecast, but it can hardly equal in importance the provision of walls and roof built to minimize the escape of warmth in winter and the penetration of heat in summer. The average citizen in many parts of the United States, if not elsewhere, would long since have objected to the relatively poor housing offered him had he been aware that its defects were unnecessary and out of harmony with modern progress. Indeed, some primitive peoples provided themselves with shelters better suited to their environment than those generally built for civilized man, particularly in the large communities of recently settled countries. In the past, adequate heat insulation was perhaps obtained in structures of the better class, if only incidentally, by reason of the thickness of walls and general stoutness of construction employed, because of ignorance or lack of stronger materials, or for purposes of defense. Today, without unnecessary waste of building materials, and at modest expense, dwellings can be provided that will house the tenant comfortably, irrespective of the seasons, and permit fuel economy in winter. Rapid replacement of the numberless inefficient houses already built is not to be expected, but the trend is under way with respect to new construction.

Much ingenuity has been expended in recent years in devising new and better ways of building houses. A survey conducted by a steel company is said to have disclosed more than fifty different construction types, of which less than six contributed to greater economy and improved service—a circumstance that is surprising because the fundamentals are understood and the materials available.

Among the materials most used to secure heat insulation are gypsum, asbestos, limestone, and diatomite. Superior to most other insulating materials because of their inertness and resistance to decay, these nonmetallic mineral products are sure to find a widening market. Still another efficient insulating device—one of the newest—is metal foil, crumpled in a dead-air space.

Gypsum is freely used in one of the most recent types of housing construction to be developed. This type, said to be entirely practicable and relatively cheap, is described as consisting of a steel structural frame of the skeleton type, composed of columns, girders, beams, and rafters without studs, joints, or other parts used in wood construction; and having exterior walls, partitions, floors, and roofs of prefabricated panels or units of suitable size and adequate strength, securely attached to the structural frame. The core or body of the panels is made of aërated gypsum, finished on the inside with plaster board and having a weather-proof exterior finish, with floor and roof surfaces finished as desired. The panels for the vertical parts are of story height, the floor and roof panels of sizes suitable for the arrangement of the supporting steel work. Windows and doors are incorporated in the panels. Joints are so made that the passage of air and heat is prevented. This method of construction is said to provide a structure comparatively low in cost, quickly erected, and proof against fire, cyclone, earthquake, vermin, and weather, as well as insulated against the transmission of temperature and sound. Construction can be adapted to all styles of architectural design and all types of heating, lighting, and plumbing apparatus. Such a structure can be dismantled and re-erected upon another site at small cost.

This method of construction furnishes an example of the intelligent application of known principles and readily available materials to the housing problem. Dwellings thus built would be superior to those erected in many mining camps, even in some where money has been freely spent. One may hazard the opinion that much housing of the future will be of this or some related type.

Trends in Practice First Spread Locally

A DECADE AGO were published the details of a change in power-plant practice that resulted in significant economies and simplification of operations. The Lonely Reef Gold Mining Company, operating in an isolated district in Southern Rhodesia, had faced for several years before the increasingly difficult problem of obtaining fuel for a boiler plant that provided steam for power purposes. Available fuel supply became more and more distant; and transport, especially in the wet season, was expensive. Many phases of plant superintendence and control became complicated, and results showed a low mechanical efficiency. Costly features of operation included the need of pure water for the boilers, or additional maintenance expense; with the necessary condensing equipment the power plant occupied much space.

A change in practice was put into effect that resulted

in a decrease in fuel consumption from 3,000 to 600 cords per month, a reduction in operating expense of £3,000 per month, and a welcome diminution in labor required. The scattered boiler, steam-engine, and condenser plant was discarded and replaced with four compact gas-producer units and an equal number of gas engines. Three of these were connected direct to generators, and the fourth to a compressor. The makers guaranteed a fuel consumption of no more than 2.15 pounds of dry wood per brake-horsepower per day, but operations indicated a consumption of only 2.07 pounds. The change from steam to producer-gas power resulted in a saving of about ten shillings per ton of ore milled, and many other advantages were recognized. A fifth unit was subsequently added.

The effect of this innovation in local practice has been striking, sixteen mines in the Rhodesias being now equipped with forty-three producer-gas units. Eleven of these operate on bituminous coal, three on bituminous coal or on wood, and twenty-nine on wood. The most significant feature of the innovation is that it has proceeded without setback. Users of the newer form of power generation equipment are more than satisfied with the change; and tribute is therefore due to the designers and manufacturers of producer-gas plants for their contributions to the solution of an important problem of mine operation in an isolated district—the provision of a steady source of power at a reasonable cost and without dependence on a large labor force. Another important advantage in favor of the producer-gas plant in regions where wood is the only fuel available, or is the cheapest fuel, is that reserves are not wasted. Many complications in power-plant superintendence and control are obviated, because a simple technique of operation has been developed that is available to those who are inexperienced in the use of the internal-combustion engine as a prime mover.

When the results of the trend in power-plant practice in the Rhodesias are better known, an opportunity existing for the introduction of producer-gas plants in other parts of the world will be realized, and steps will be taken to effect economies similar to those achieved at the Lonely Reef and at many other mines in Africa.

More Light on Wet Extraction Processes

THE NEED FOR INFORMATION on the mechanics of the simpler processes of extracting a soluble metal associated with an insoluble gangue, at a time when effort is being made to foster activity in profitable gold-mining enterprise, justifies renewed discussion of a subject that was brought to the attention of metallurgists last year: the high-extraction and low-cost results obtainable in the treatment of very fine sand by gravity leaching. Assays of sieved samples of a typical sand charge at the Homestake prompted attention to the failure of so many technicians to realize that, although a comparatively high percentage of amorphous slime may not interfere with the leaching of ore that has been dry-crushed and evenly distributed in a vat, the practice of grinding in water or in cyanide solution liberates such slime and causes it to agglomerate to an extent inhibiting its even redistribution among the coarse crystalline particles. Thus, when mill pulp is delivered direct to a leaching vat, the settled sand invariably contains seams of slime that cause short circuiting and imperfect extraction.

The necessity and economy for thoroughly de-sliming an ore pulp by the use of classifying apparatus, whereby a sand is produced that is uncontaminated by clay or other colloidal material, has been demonstrated. At the Homestake, the battery pulp is carefully classified into crystalline material and amorphous material. The physical character of the product, not the mesh size, marks the line of demarcation. Amorphous 200-mesh slime is kept distinct from 200-mesh sand. As a final precaution, the classified sand is delivered to a leaching vat filled with water and provided with an annular overflow launder so that no contamination of the charge with seams of non-crystalline fine material is possible. Metallurgical results, in consequence, are worthy of note, particularly in regard to the high extraction obtained—90 per cent—by means of simple gravity lixiviation, from the minus 200-mesh material. This find sand was shown to consist of one-third, by weight, of the charge; so that, based on current production figures, about 260,000 tons is being treated per annum, at low cost and with high recovery.

Laboratory research has since been continued, in an effort to disclose further facts. Two new testing screens were used—260 and 325 mesh, respectively. With these the fine material was again classified, the results showing that no less than 85 per cent of the minus 260-mesh plus 325-mesh product is a sand, treated as such at rest by gravity percolation, and offering no obstacle to the even percolation of solution and the successful extraction of the gold.

Contrary to an impression that is widely held, the operation of each phase of treatment procedure in the lixiviation of a charge of sand is not ascertainable by visual examination of externals. Fundamental research is needed. This may be of a type that appears at first as of academic interest only; but the results invariably indicate, as they have done at the Homestake, the economic and technical soundness of a simple metallurgical process, as well as providing evidence of great help when decision has to be made in respect to the imitation of technical trends at other properties.

Work at the Homestake may well serve to indicate a desirable program of research and experimentation to determine the best and most economical method of treating a gold ore of similar type and grade. It suggests that, with other types of ore, particularly those that respond to flotation treatment, the maximum of efficiency and economy will be secured by the steady acquisition of technical knowledge, supplemented by a policy of frank publicity in regard to plant results, economic as well as metallurgic. This exchange of information has made possible the success of cyaniding in the last thirty years. A similar spirit of cooperation among the present generation of metallurgists will bring a complementary tool, flotation, to successful and extended use in the econmical and efficient treatment of gold ores.

Application of Geology to Mining

Harrison Schmitt

IN RECENT YEARS a few entrepreneurs and geologists have come to realize the possibilities of a certain type of detailed geological mapping for finding new ore, when applied particularly to areas in which ore replaces limestone and where ore deposition was governed by structural features of the country rock. For example, when a diamond-drill development campaign in a certain limestone replacement camp in the West was started without geological control, no ore was found for the first six months of operation. Geological control was instituted during the seventh month—the geological department was then given charge of all exploration and development work—and from the tenth to the fourteenth month 380,000 tons of blind ore was found as eight separate orebodies at a cost of $0.27 per ton. Previous costs had been from three to four times as great.

The structure was worked out by following and mapping key beds with the plane table as in oil-field mapping. Primary transit surveys were used for control, and the map scales varied from 1 in. = 100 ft. to 1 in. = 30 ft. In addition, gossan and metamorphic halo studies were made, with the result that well-defined criteria for hidden ore appeared. Geological work of similar intensity and on similar ground in an old district in Utah where the ore was blind—covered by 800 to 1,200 ft. of barren ground—resulted in the discovery of several new mines. The results of these two tests and several others demonstrate that detailed technique in limestone replacement camps will result in the finding of much new ore. In such terrane geological work can be precise and convincing.

The best type of mapping and geological observation for a particular piece of work is often difficult to select. Usually, the outcome or magnitude of the usefulness of geological work cannot be foreseen, which puts the geologist at a great disadvantage when he is asked to plan a survey and estimate the appropriation needed. The object of this paper is to suggest methods believed to be appropriate for the various types of work.

Vein Deposits—Until recently, geologists were forced to admit that geological work on veins was of little value unless problems of faulting, such as those of the veins at Butte, Mont., were encountered. The miners are in their element when following the developing veins, and have needed little guidance. Recently, however, several classes of technique—now in various stages of development—have appeared, which make the development of veins more efficient,

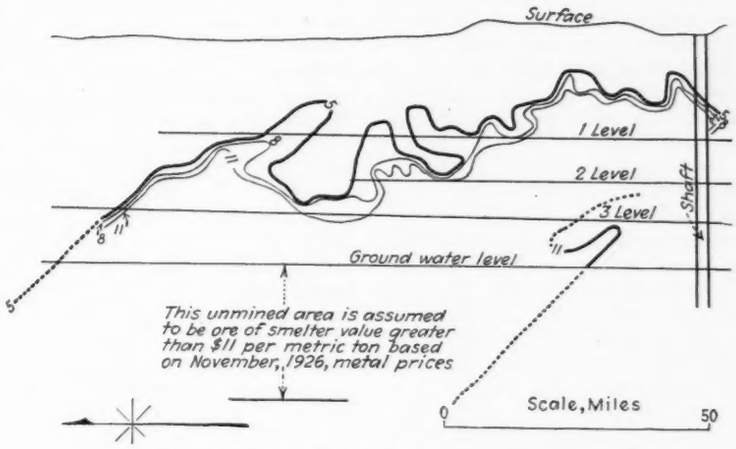

Fig. 1—Equal-ore-value lines of an oreshoot on longitudinal vertical projection. The interval is $3 per metric ton.

and aid in finding blind oreshoots. One of these techniques has to do with the horizontal and vertical extension of oreshoots (Schmitt, Ref. 5). Oreshoots are outlined by various quantitative methods, and predictions of extensions made from purely mechanical bases (Fig. 1). From a more strictly theoretical point of view an attempt is being made to develop criteria for true and apparent bottoms of veins and oreshoots. We hope that we shall be able to distinguish between the fundamental downward playing out of the mineralization and a mere local pinching. Some advance has been made on this problem. For example, in the Santa Barbara district, Chihuahua, Mexico, a coarser texture of the ore and gangue minerals in depth means the true bottom of the ore. In the Red Cliff district, Colorado, the appearance of pyrite in the form of a chimney apparently marks the end and bottom of the manto orebodies. Other criteria of similar nature are becoming known.

Another technique is concerned with the location of blind oreshoots along the vein outcrop (Lausen, Ref. 2, pp. 88, 124-25). Apparently, many blind oreshoots have low-grade "leaks" that extend to the outcrop (Fig. 2). Usually exhaustive macroscopical and microscopical studies are made of the known blind oreshoots that are expressed by such "leaks," in order to reveal the normal outcrop expression, with the hope, of course, of reversing the process in unknown ground and finding new ore. Commonly, a gangue mineral resistant to weathering, such as a certain variety of quartz, fluorite, or barite, is found that is genetically and spatially related to the ore minerals. The ore minerals usually leach from the outcrop—if there is sufficient pyrite, even a gossan may disappear (Locke, Ref. 3)—so the residual minerals may be accurate criteria for ore below.

Gossan and capping technique has been worked out and described by Locke (Ref. 3), Boswell and Blanchard, (Ref. 4), and others. It merges with the technique just described; it has become invaluable in the studies of vein outcrops and the cappings of disseminated copper ores, and needs no elaboration here.

When studying veins, it is unnecessary to map in detail and on a large scale more than narrow strips of ground along them, unless the orebodies are localized by broad structural features of the country rock—an uncommon occurrence. The relation of vein to vein in a district can usually be shown on a relatively small scale, such as 1 in. = 1,000 ft. This is, of course, in sharp contrast to the large-scale maps needed in structural mapping in areas of limestone-replacement deposits. With veins, as a rule, the emphasis is put on large-scale underground level plans from which cross-sections and various types of longitudinal projections of the vein and oreshoots can be compiled.

Bedded Deposits—Geology, when concerned with bedded deposits such as coal, salt, and iron, approaches engineering in the accuracy of predictions based upon it, and is particularly useful when fault problems are to be solved. The depth to a given bed can be predicted, even in strongly faulted country, with an accuracy of 5 per cent down to 500 ft. and 10 per cent down to 1,000 ft. when the control is a plane-table map on a large scale, such as 1 in. = 100 ft. The procedure is essentially that used in oil-field mapping—sections are meas-

ured, key beds are followed, and other and complementary work is performed.

Disseminated Deposits—Recent discoveries of primary copper ore, some of it high grade, below low-grade disseminated copper ore at Ely, Nev., and Cananea, Sonora, have changed our point of view on disseminated orebodies, hitherto considered as largely engineering problems. The new deep ore is associated with structural features of the country rock, and so further investigation of this type can use geological tools to advantage. The procedure in order is large-scale mapping, with emphasis placed on the measurement of structure.

Replacement Deposits in Limestone—As discussed in the introduction, one of the most profitable fields in which geology has been applied is that of replacement deposits in limestone where the deposition of ore has been controlled by the structure of the country rock. Large-scale mapping is supplemented by studies of metamorphic and alteration halos, gossans, cappings, and "leaks." A tested and effective procedure is to begin with a careful measurement of the stratigraphic section, paying particular attention to key beds, and to follow this by plane-table mapping on large scales, such as 1 in. = 200 ft. and up to 1 in. = 30 ft. (Schmitt, Ref. 6). In complicated ground, experience has shown that the structure should be measured first. Then the various special studies should be taken up on the same ground. When the type of country-rock structure which localizes ore and the earmarks of the ore, such as alteration, "leaks," and gossans, have been made out, similar criteria should be looked for in the undeveloped ground.

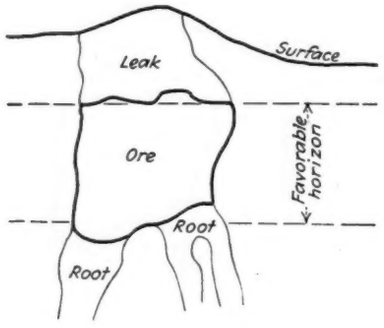

Fig. 2 — Longitudinal vertical projection of a vein showing orebody, roots, and "leak."

Examination of Prospects—The type and amount of work needed in the examination of a given prospect will vary, depending upon such factors as the amount of development, type of deposit, and object of examination. Joralemon (Ref. 1) shows that the amount and quality of the work may be determined by the "weakest link"; that is, that an obvious factor such as lack of water, geographical location, weakness of mineralization, shows the prospect to be valueless, and further examination a waste of time. Opposed to this idea to some extent is the need for permanent and more or less complete data on prospects by a company or an individual actively searching for ore, so as to avoid reexaminations because of changes in the value of prospects as the result of changes in metal prices, metallurgical processes, construction of new roads, canals, or other factors. Besides, although the examining engineer may be fully reliable, managements are demanding recorded facts to support his opinions. In the Southwest, few reconnaissance examinations have resulted in more than a letter to headquarters as a report. Maps have rarely been submitted. The result has been that some prospects have been examined again and again. After a year or so even the engineer who made the examination no longer had a clear picture of the conditions, or he may have left the company. One company examined a zinc prospect in Arizona four times in sixteen years. None of the opinions in the first three reports could be judged by the data offered. To correct this real difficulty I suggest not that more time be given to each examination, but that the engineer plan the work ahead and with reconnaissance equipment make maps and cut as many representative samples as time permits.

A mistake often made is that of sending an inexperienced man on examination work. Long training alone gives the faculty of quickly determining the structure of the country rock, the importance of the mineralization, and the tonnage, grade, and possible metallurgical treatment. The ground must be walked over anyway, so the engineer or geologist may as well carry a stadia rod and locate his points of observation, few and incomplete as they may necessarily be. In this work an assistant is needed to operate a plane table and alidade, and a companion is usually necessary in and around the mine, in case of accident. If no assistant is available, the surface mapping can be carried out with a traverse board and open-sight alidade, which will give a satisfactory preliminary map, but is slower than the standard plane table and alidade method. Only two days should be needed to map an area several thousand feet on a side, survey five hundred feet of mine opening, and cut twenty or more samples. This information should be sufficient to support a report of the type indicated as desirable.

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Colorado Alumni Make Leases Pay

ENGINEERS LEASE, INC., sponsored by the Colorado School of Mines Alumni Association, is operating leases on sections of the Vindicator and Last Dollar mines, near Victor, Colo., writes H. L. Minister. At the Vindicator property, a stope 100 ft. long, containing $14 ore, has been opened. As the ore continues in the breast of the drift, the stope will probably be much longer than 100 ft. when it has been fully developed. About 40 mine cars of ore, of 1,400 lb. each, are hoisted daily, which is the maximum amount that the shaft can accommodate from any one lessee. Prospects at the Last Dollar mine are also encouraging, although a definite orebody has not yet been discovered. Several carloads have been shipped that represent ore encountered in development.

In addition to giving employment to young graduate mining engineers during these stringent times, Engineers Lease has afforded these men practical experience under highly favorable conditions. Operations have been so organized that each man takes his turn at various jobs. A year's service with the "Lease" gives him a practical education that is usually not obtainable in college course. He learns how to handle a rock drill by operating one underground; he learns how to build chutes and to place timbers; he acts as foreman of one of the leases and calculates costs; he serves on the "Board of Strategy," doing sampling, making geological studies, comparing working maps, and formulating plans for the future of the company. From the time that he becomes a regular employee he attends weekly meetings where all the problems and business of the company are discussed. Consequently, he becomes familiar not only with the technical problems of the company but also with those of management that confront every mine operator. In short, Engineers Lease is providing, on a financial basis particularly favorable to all concerned, an exceptional post-graduate course for a number of the alumni of the Colorado School of Mines.

Flotation-Cyanidation Supplants All-Cyanidation in Russia

M. C. Haas

KOCHKAR gold mines, about 50 miles southeast of Chiliabinsk, in the southern Ural Mountains in Russia, are situated in what formerly was a placer district. Two foreign concessions operated there, one French and one English. The ore was not high in grade, but cheap labor and selective mining insured profitable operations prior to the revolution, when work ceased. The properties were later taken over by the new government, and plans made to rehabilitate them. The power plant was improved and enlarged to make possible a larger output; a new machine shop was erected, and the mine equipped with modern American machinery. On the French concession a 300-ton all-sliming cyanide plant had been built in 1911. This was remodeled. Ball mills replaced Chilian mills and tube mills, and Oliver filtration was adopted. Tonnage of the new mill was to be 600 per day. By the spring of 1930 the work had been completed, and the property was once again placed on an operating basis. Numerous difficulties were encountered, however; interruptions to output occurred, and tonnage treated was only a small proportion of the plant's rated capacity.

The ore was divided in two classes: that from the North Group was clean quartz, occurring in veins averaging 0.7 meter wide and having a gold content of 5 to 10 grams per ton. On the hanging wall, however, was a vein of schistose material varying from a few inches to 2 or 3 ft. wide. Sulphide content of ore from this group averages about 2 per cent iron, as FeS_2. In the South Group the vein material is mostly schistose, with some quartz, with a content of 5 per cent and more iron, as FeS_2. The grade of this ore was 4 to 5 grams gold per ton.

Chemically, the ore was ideal for cyaniding. Aside from 1 to 5 per cent iron and a trace of arsenopyrite, no other sulphides or oxides occurred in quantity sufficient to cause difficulties. However, the presence of excessive amounts of schistose material complicated thickening and filtering operations. By careful mining, the French had been able to keep the proportion of such material down to 15 per cent, thereby minimizing trouble. Under Russian supervision the decision was made to adopt cheaper mining methods, which resulted in a much greater dilution of the ore with schist. Development of South Group ore, with a high schistose content, resulted in a mill feed containing 50 to 60 per cent of this material. Dependence was placed on hand sorting for its elimination.

The original mill was erected on flat ground, no more suitable site being available. In remodeling the plant the Russians utilized the existing mill building, which was a well-built frame structure, and added to this as required. A flowsheet was developed from past practice in the district. The mine flooded after the shutdown, at the beginning of the revolution. Mill construction could not be delayed until the mine was unwatered and samples procured for settling and filtering tests and for checking metallurgical results. The method adopted, a combination of past practices in the district, was to grind to minus 35 mesh, treat the pulp by plate amalgamation, then classify into sand and slime. The slime was thickened, agitated in Pachucas, and filtered in Oliver filters. The sand was passed over Wilfley tables to remove the sulphide for regrinding. Table tailing was treated on sand filters, then conveyed to leaching vats.

The following details provide a description of the new crushing plant as it will be when completed, and the remainder of the equipment as it was when placed in operation: Ore coming from the South Group was hoisted directly to the coarse-ore bin; ore from the other two shafts was transported in cars drawn by steam locomotives. These loaded cars are to be hoisted from the track level to the coarse-ore bin. Grizzly bars with 16-in. spacings will be placed over the bin; oversize is to be broken with sledges. A 30-in. Russian-made apron feeder will deliver the coarse ore to a second grizzly with 3½-in. spacings. The undersize drops onto an inclined conveyor, and the oversize is fed to a 16-in. McCully gyratory crusher set to 3½-in. and driven through Texrope by a 100-hp., 712-r.p.m., 250-volt slip-ring motor. Crushed material joins the grizzly undersize and is conveyed to the sorting plant.

No. 1 conveyor will discharge into a 48-in. trommel screen, the upper section with 1¾-in. perforations and the lower with ¾-in. perforations, and equipped with a spray for washing the ore before sorting. Undersize from the first section, which will represent about 35 per cent of the total, drops onto an 18-in. conveyor leading to the secondary crusher. The plus 1½-in. and minus 3½-in. product will be split between two 30-in. sorting belts having a belt speed of 50 ft. per minute. The waste sorted from the ore is dropped into hoppers directly below the sorting belt; when these are full the waste is to be drawn into cars and trammed to the waste dump; 30 per cent of the original ore is to be sorted out as waste. Water and slime resulting from washing the ore prior to sorting flows to dewatering cones, the dewatered pulp passing into the regular mill circuit.

Undersize from the trommel will then pass over a vibrating separator with ¾- or 1-in. screen, the undersize dropping to another inclined conveyor. The sorted ore and the oversize from the vibratory separator will be fed into a 3-ft. Symons cone crusher driven by a 50-hp. motor through belt drive. This crusher is expected to handle 40 tons per hour, to ⅝ in. The crushed product then joins the screen undersize and is conveyed to a 600-ton flat-bottomed wooden bin. A Russian-made tripper will be used for distributing ore into the bin.

Coarse crushing is now being performed by a Gates gyratory crusher taking a 10-in. feed and crushing to 2½ in. The fine ore is drawn off by four 24-in. Stephens-Adamson belt feeders, with a belt speed of 10 ft. per minute, and driven by 3-hp. motors through S.A. worm-gear speed reducers and chain drive to the head pulley. The feeders are arranged in sets of two, each discharging onto a 16-in. conveyor, which in turn discharges into the ball-mill feeder. Each 16-in. conveyor has a Sintering Corporation weightometer.

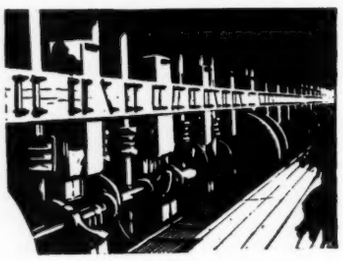
Fine grinding is performed in two No. 75 Marcy ball mills, directly connected to a 150-hp. 2,000-volt, 360-r.p.m. induction motor, using Falk-Bibby couplings and herringbone gears. Each mill is in closed circuit with a 6 ft. x 26 ft. 8 in. Model DSD Dorr classifier, set with a slope of 3 in. to the foot and operating at a rake speed of 25 strokes per minute. The Dorr classifiers are driven by 5-hp. motors through Whitney, silent-chain drive. Each mill, charged with 18,000 lb. of forged-steel balls, uses 4-in. balls for replacement. The grate openings are ⅜ in. Ball consumption varies between 0.8 and 1 kilo per metric ton of ore milled. Classifier-overflow pulp density averages about 48 per cent solids, overflowing a product containing 2 or 5 per cent on 35 mesh and 35 per cent minus 200 mesh. Ball-mill discharge is carried at 65 per cent solids. Because of the high density of classifier

overflow, circulating load is low—probably 150 per cent. Each mill can grind 300 tons of 2½-in. mill feed per 24 hours, if milling conditions are watched.

Classifier overflow is laundered to a Wilfley pump and elevated to the amalgamation floor. Here the pulp is automatically sampled and then flows to two Coeur D'Alene pulp distributors, each feeding twelve 48-in. x 12-ft. copper plates—2 sq.ft. of plate area per ton of ore. The plates are set with a slope of 1¾ in. per foot, and the pulp is diluted to 75 per cent water. Each plate is supplied with a small trough-type trap, and the amalgamation tailing launder is lined with coco matting, which picks up much mercury. This launder discharges into a locally designed amalgam trap, from which the pulp flows to the second Wilfley pump. The plates are covered with sealed wire-netting frames, the amalgamator only being allowed to break the seal.

The second Wilfley pump elevated the pulp to a 4½-ft. Allen sand cone, the underflow going to a distributor and to three No. 6 Wilfley tables, riffled for 48-mesh material and driven by a 1½-hp. motor through a two-strand Texrope drive. The sand cone overflow is split between two 6-ft. Allen slime cones, the underflow being distributed between five more No. 6 Wilfley tables riffled for 80-mesh material. Concentrate from these tables is combined and elevated to a 16 in. x 6 ft. Hardinge ball mill in closed circuit with a 4-ft. Model C Dorr classifier, where it is slimed and delivered to the slime department. The small amount of concentrate inhibited continuous operation of this unit, so it was allowed to accumulate and then milled in batches. Sands from the tables were combined and elevated by a Wilfley pump to an 8-ft. Allen sand cone, the underflow from which was the finished sand-plant feed, represented between 40 and 45 per cent of the initial feed and had a moisture content of 30 per cent. Further dewatering of sands was accomplished by means of two 11 ft. 6 in. Oliver rotary sand filters, each having a rated capacity of 125 to 150 tons per day and making a product containing 17 to 23 per cent moisture, depending upon the character of ore milled. Each sand filter elevated its discharge to a 14-in. conveyor, from which the sand was charged into any one of six 150-ton vats by means of a S.A. tripper. The filters and conveyors were driven by 3- and 5-hp. motors through S.A. worm-gear speed reducers and chain drive. One man was required to level off the sand when the vat was full. The leached sand was shoveled out through four openings in the bottoms, costing 12c. per ton for labor, each man handling six tons per shift. The sands after loading into cars were elevated by inclined tramway to the sand dump. A Wilfley pump is on the ground and plans are ready for introducing hydraulic discharging in the near future.

Overflow from the 8-ft. sand cone and the two 6-ft. cones is combined and laundered to two 30-ft. Dorr thickeners. The slime represents 55 to 60 per cent of the initial feed. Ordinarily, 1 to 3 per cent remains on 65 mesh, and 65 to 68 per cent passes 200 mesh. Thickener feed has a water-solid ratio of 10 or 12 to 1, and the underflow is 1.2 to 1. Thickened pulp flows to a 30-ft. collector having a capacity of 70 tons of dry slime. Here NaCN is added in the proportion of 400 grams to the ton of ore, giving a working solution of 0.05 per cent in terms of KCN. These batches are charged intermittently into three 15x45-ft. Pachuca vats. Agitation is for eighteen hours at 22-lb. air pressure, after which the content is discharged into a second collecting vat. The agitated pulp is then elevated to three 8x14-ft. Oliver filters driven by 5-hp. motors through speed reducers and chain drive. Cake is washed with the usual spray attachments. Residue is discharged into a launder, sluiced into a 25-ft. repulping tank, and pumped to the tailing dam by a 6-in. sand pump.



The effluent for the sand plant is collected in a small storage tank and elevated by centrifugal pump to two clarifying tanks having porous bottoms, using leached sand as a filtering medium. From here it flows to four ten-compartment zinc boxes. The Oliver filtrate, the only solution from the slime department that is precipitated, is handled in similar manner, being delivered to another set of four ten-compartment zinc boxes. The zinc shavings are dipped in a solution of lead acetate before charging to the boxes, and sometimes a small drip of cyanide solution is used. Precipitation is good. Occasionally, considerable white precipitate forms, but by watching the alkalinity and cyanide strength, and with frequent dressing, this does not become serious. Zinc consumption is 200 grams per ton of ore treated. The Merrill-Crowe process is to be installed as soon as funds are available.

Zinc boxes are cleaned every two weeks, the precipitate is treated with weak sulphuric acid to remove the zinc, and the sludge collected and filtered, in linen bags, then roasted in rectangular trays, broken up and mixed with flux and smelted in oil-fired pot furnaces. A bullion assaying about 800 fine in gold is produced.

From a metallurgical standpoint the ore offered no serious problem. In the amalgamation department, although the recovery was not as complete as might be expected, the losses were due to fine gold difficult to amalgamate but readily recovered by cyanidation. About 60 per cent of the total gold is in the native state, but only 35 per cent of the total is recovered by amalgamation. Milling in cyanide solution resulted in the formation of an extremely hard amalgam. Plates were dressed and cleaned four times a day, using a weak solution of cyanide. Caustic soda and ammonium chloride are also used. Care must be taken in the use of cyanide if the plates are to be kept bright and in good condition. Amalgam amounting to 1.5 to 2 grams gold per ton was removed once a day, pressed by hand in linen sacks to remove the excess mercury, cleaned in a mortar, and retorted in an oil-fired furnace. Mercury consumption is 3 grams per ton of ore. The amalgam contained 38 per cent gold. The resultant bullion assayed 900 fine.

Inasmuch as 40 per cent of the gold was associated with the sulphide, to remove as much of it as possible from the sand plant feed was imperative, as aëration and contact was not active enough to recover the gold from the sulphides. This the tables did fairly satisfactorily, although the greater percentage of loss was in the sulphides. Recovery by percolation was good as soon as the proper conditions had been determined. The sand offered no serious difficulty in percolation, the rate amounting to 3 in. per hour; 3 to 8 per cent of the sand remained on 35 mesh, and 8 to 10 per cent passed 200 mesh. Best results were obtained by using 25 to 30 tons of strong solution (0.12 per cent) to a 150-ton charge, allowing it to stand for 12 hours, then draining it and charging in another 25 to 30 tons of solution of the same strength, allowing it to stand for 12 hours, draining, then adding 75 tons of weak solution (0.04 per cent), allowing it to stand for three to four days, following this with wash solution, making in all a six-day cycle. Effluent is drawn into a storage tank, where it is checked for cyanide content, then pumped to the clarifiers.

Good extraction was possible in the slime plant only by using 0.05 per cent solution and 18 hours' agitation. The only difficulty encountered was inability to dewater and wash the agitated pulp.

The flowsheet described in the foregoing refers to the original mill. The plant, disregarding insufficient thickener capacity, would have been satisfactory had the ore been an average clean quartz. However, the change in mining methods and the development of new orebodies carrying larger percentages of schist involved difficulties in filtering and thickening that could not be over-

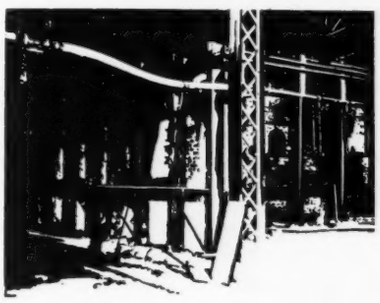
come with the equipment at hand. Sorting is expected to alleviate this trouble; but with 35 per cent of original ore bypassing the sorting section as fines, which will naturally carry a large proportion of schist, and because of the development of the new orebody in which the schist carries gold, a clean quartz feed cannot be expected.

Presence of the schistose material increased thickener area requirements from 4 to 6 sq.ft. to 12 to 15 sq.ft. Settling tests made on slime-plant feed indicated a 12 to 18-sq.ft. area requirement, according to character of ore delivered, when making a 1.2 to 1 underflow product, which varied according to the amount of schist and decomposed granite in the ore. Filter capacity for several months never exceeded 100 lb. per square foot of filter area per 24 hr., and the cake was of such a character that washing was impossible. This fact reduced mill capacity from a rated output of 600 metric tons down to scarcely 200 tons per day. The pulp was so highly dispersed that much lime was necessary, and even then a pulp condition could not be secured that permitted a higher capacity than 200 lb. per square foot of filter area per day, or 35 metric tons per filter-day. A combination of the colloidal decomposed granite, flat pieces of micaceous material, and excess lime blinded the filter cloth. When using much lime the cake became soggy, retaining a high moisture content, and inhibiting washing. Many remedies were tried. Fine-weave filter cloth gave better results, probably because the small openings did not clog so readily. Double-cover combinations were tried, without improvement. Drum speeds were varied. The lower front quadrant of the filter valve was blocked off on the supposition that the coarser sand would be classified and picked up on the drum first, thus avoiding the clogging of cloth with slime. This did not help. Coarse sand was mixed with the filter feed, resulting in satisfactory filtration, obtaining capacity and effective washing. However, to make a practice of this would have been out of the question. By delivering a pulp of a maximum density allowed by limited thickener capacity, by scrubbing and washing the filter covers with acid frequently, and by controlling alkalinity closely, we were able to bring the capacity up to 200 lb. per square foot of area, and no more. By using only five or six spray nozzles, and by inverting them so that only a fine spray fell on the cake, we were able to obtain some displacement without excessive amounts of wash solution running back into the pulp. Cake thickness averaged between $\frac{1}{8}$ and $\frac{1}{4}$ in., which with an ore of 2.6 sp.gr. indicated low filter capacity. I tried changing the flowsheet around so that one thickener could be used for giving a water wash to the agitated pulp before filtration. This, of course, lowered the gold content of the solution in the filter pulp and lowered our dissolved-gold losses, but thickener capacity was so small that normal tonnage could not be handled thus.

Thickener capacity was evidently far too low, and after a few months' work the need for radical change became evident if 600 tons per day was to be milled. To install enough filters to handle the 350 tons coming to the slime department would be out of the question. Attempts had been made to grind to a coarser mesh, and thereby divert more sand to the sand plant. This did not prove practicable, because of resulting losses of gold in the coarse-sand tailing. Counter-current decantation without filtration would have been particularly well adapted to this type of low-grade ore, but the tank capacity required would have meant an expensive addition to an already expensive plant, involving heavy housing and heating charges.

Soon after I arrived and became acquainted with the problem I started campaigning for permission to determine the possibilities or introducing flotation as a method of concentrating the gold before cyanidation, and permitting the rejection of the refractory material causing the trouble. I had difficulty in making my point. The change would take several months. The outcome was doubtful, but something had to be done to keep mill capacity ahead of mine production, which was about 300 tons per day.

We decided to resurrect the Butters equipment. The French records showed that the filter formerly gave a capacity of about 2 tons per 5x10-ft. frame, with low residue loss. The French, however, were treating an ore carrying only 15 per cent of schistose material, and sliming all of it. We had 50 to 60 per cent, and with the sand-slime separation in vogue the greater part of this material went to the slime plant. Consequently,



before going ahead, we rigged up a 2x3-ft. Butters frame and tank, and made tests on slime-plant pulp. The results of these tests indicated that satisfactory results could be obtained with the Butters filter, with probably a 20 to 25 per cent lower capacity than formerly obtained. Ninety Butters frames were immediately installed and placed in operation, with satisfactory results. Each frame had a capacity of 1.5 to 1.75 tons. Washing of the cake was such that soluble losses were less than a 0.1 gram, and total residue losses 0.5 to 0.6 gram on 5-gram ore. This seems to be a case where the intermittent filter, which allows long vacuum and washing periods, is suitable.

When the sand filters were ordered, preliminary plans were made to leach a minus 10-mesh, plus 48-mesh sand; later those responsible for operations decided that finer grinding was necessary, and the question came up as to whether or not the sand filters would handle this material. A trial was finally decided. The character of the ore was such that the canvas quickly blinded, and filtration would stop. Various classes of material were tried, with no help. Coarse sand beddings were placed over the cloth, but this also allowed solution to accumulate in the filter tank. Finally, by an arrangement by means of which excess water could be siphoned off, we were able to operate them. Reduction in moisture, however, was not sufficient to warrant their use, but it permitted use of the conveyors for distributing the sands to the vats, and avoided the necessity of making other arrangements. A Dorr classifier was installed beside one of these filters and given a trial, with fairly good results. A product of about the same moisture content was produced, which could be distributed by belt conveyor.

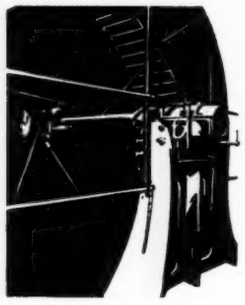
With a clean quartz ore, and recovery as obtained by straight cyanidation, no justification could be seen for introducing a different treatment. However, serious mechanical difficulties arose in both departments, requiring heavy expense. Consequently, I was given permission to take a representative sample of the ore to the Ginsvetmet ore-dressing laboratories in Moscow. Results from this work indicated that a high recovery by flotation was possible at the same degree of comminution as that now practiced, with a ratio of concentration averaging 35 to 1. This meant the rejection of 97 per cent of the total ore, leaving only 2 or 3 per cent for cyaniding. An appropriation was allowed, to erect a pilot mill to verify laboratory results.

Because of the premium on gold, recovery by flotation had to approximate that of cyanidation, regardless of economies in local commodities. This recovery should preferably be obtained by the same degree of comminution as for cyanidation; cost of steel grinding balls, mill liners, and power were excessive, and would offset expected economies should fine grinding be necessary. Lastly, the gold had to be recovered in a concentrate amenable to cyanide treatment. If flotation could do this, it offered the advantage of eliminating some thickening and filtering steps, with a further advantage of decreasing operating costs about 15 per cent, and all at an expenditure that would be less than the require-

ments to execute a full-tonnage program on the existing flowsheet basis.

Laboratory results indicated that a recovery of 93 to 95 per cent on a 7-gram ore was possible by an amalgamation-flotation treatment, 55 to 60 per cent of the gold being recovered by amalgamation, the remainder by flotation. Tabling of the tailing after flotation did not increase the recovery. Flotation produced a concentrate containing 160 to 200 grams gold, at a ratio of concentration of 35 to 1, with a tailing containing 0.4 to 0.5 gram gold. These results were obtained when grinding to 40 per cent minus 200 mesh and using 0.15 lb. ethyl xanthate, 0.15 lb. pine oil, and 2 lb. soda ash. Laboratory results were not improved by the use of sodium sulphide, and its use might introduce difficulties in the re-treatment of the concentrate.

A decision was made to carry on further experimental work on a pilot-mill basis, rather than to accept laboratory



results as final. To reproduce plant grinding conditions in the laboratory is difficult; the finer-mesh products gave low assays, with the losses in the coarser products, so data were necessary to determine what these losses might amount to under plant grinding conditions, when the proportion of fine- to coarse-mesh products is lower for any given limiting mesh.

Laboratory amalgamation tests gave a high recovery. In practice, however, complete recovery of the free gold was not possible, because of its fineness; and the question arose as to how completely the remainder of the free gold could be recovered by flotation. Tests made without prior amalgamation indicated that a recovery of 83 per cent was possible, which meant that about 10 per cent of the recoverable gold was unfloatable.

Further pilot-mill testing was advised to check the amenability of all classes of ore delivered to the mill. Shrinkage systems of mining were practiced, which meant the possibility of partial oxidation of the pyrite and arsenopyrite. Aside from flotation machines, sufficient equipment was available to build a pilot mill with a capacity of 3 tons per hour. The 16 in. x 6 ft. Hardinge mill, in closed circuit with a 4-ft. Model C Dorr classifier, was arranged for fine grinding; pumps and amalgamation plates were already in place; only launder alterations were needed. For flotation machines I selected a four-cell No. 21 Fahrenwald, so set that use could be made of a spare Wilfley table for treating the flotation tailing. A 6-cell machine of smaller size would have been more suitable, but was not available. A locally made reagent feeder was built. Four months were spent testing this method of treatment, developing information necessary to make a comparison between the two methods.

Pilot-mill results indicated that amalgamating with fresh water developed a better condition than was experienced in the plant. The amalgam did not become nearly so hard; rubber scrapers removed it readily; the softer condition resulted in better gold collection, with much less difficulty in keeping the plates clean. By introducing inside amalgamation we were then able to increase gold recovery from about 35 per cent (as obtained in plant practice) to as high as 50 per cent by the new method. Variation of conditions indicated that results comparable with cyanide-plant recoveries were possible with the same degree of grinding, but losses in material coarser than 28 mesh were prohibitive.

Even though a mechanical type of flotation machine was used, pulp conditioning before flotation was found to be imperative, to allow sufficient contact with reagents; the low temperature of the mine water used in milling (6 to 7 deg. C.) was largely responsible for this step. A low froth column proved best, preventing opportunity for the gold to drop before being removed. The ratio of concentration is high on an ore of this type, and the presence of gangue as a diluent of the concentrate is not objectionable, so the practice of carrying a low froth column and effecting its rapid removal is desirable. The same is true with regard to circulating load: removal of a finished concentrate from as many cells as possible gives high extraction.

Poor recovery by amalgamation was reflected in our over-all results only when coarse gold was escaping, the fine gold floating readily. More difficulty was experienced in recovering the partly oxidized gold-bearing sulphides. Extraction from ore that had laid in stopes for two or three months was appreciably lower. Considerable difficulty was experienced in floating the minerals at these times, though the use of a pilot table reduced the losses. The use of sodium sulphide did not improve this condition over results obtained by the use of soda ash; and an excess did not have the deleterious effect that sodium sulphide had. Use of soda ash as a pulp conditioner and promoter—0.10 to 0.15 lb. ethyl xanthate as a collector and 0.1 lb. pine oil per ton of ore—gave the best reagent combination for sulphide flotation, and likewise gave a satisfactory recovery of free gold. The higher xanthates, the Aerofloats, and other newer reagents were not available. Undoubtedly their use would have resulted in still better results.

In starting up and operating a plant of this kind with untrained men the use of tables after flotation was recommended. They could be expected to prevent high losses in the event of irregularities in flotation operation, and would recover the coarse gold that did not amalgamate or float, as well as any coarse sulphide that did not float. In tabling laboratory test tailing, no increased recovery was obtained. However, these tests were performed on fresh unoxidized samples. Mining systems are now being considered that would avoid oxidation of the broken ore and make the use of tables unnecessary. In pilot-mill practice, when flotation conditions were satisfactory, the difference between the flotation tailing and table tailing averaged about 0.1 gram gold.

Experience in the application of flotation to this gold ore suggested that as high a recovery by amalgamation as possible should be secured, as this is the cheapest means of producing a finished bullion. Amalgamation, however, is essential to high recoveries only to the extent that the coarse and rounded gold particles be amalgamated, the fine gold in our operations being readily amenable to flotation. Good pulp agitation-aëration in the flotation machine, preceded by proper pulp conditioning, high pulp levels in the machines, and rapid removal of the froth, with uniform operating conditions in regard to pulp density and flow and reagent control, proved to be more essential than manipulation of reagent combinations. Tailing losses were lowest in the minus 200-mesh product, with but little variation up to 48 mesh. Above this, losses increased to 2.0 grams gold in plus 28-mesh material. When grinding to 35 per cent minus 200 mesh, a recovery of 88.4 per cent was obtained on 5-gram ore, losses amounting to 0.6 gram gold per ton of tailing.

Laboratory cyanidation tests on the flotation concentrate, which represented 2 to 3 per cent of the total ore and contained 130 to 200 grams gold per ton, resulted in an extraction of 95 to 96 per cent. Further refinement of methods and the intensive treatment possible for such a small tonnage of product could be expected to increase this recovery to 98 per cent or better. Laboratory practice indicated that this could be done with a cyanide consumption of 1.0 kilogram NaCN per ton of concentrate and 3 kilos lime.

Over-all extraction by flotation followed by cyanidation of the flotation concentrate equaled that obtained by all-cyanide practice. Improvement on these results could be expected under more uniform operating conditions. This, of

Metallurgical Data, Experimental and Pilot-Mill Flotation and Cyanide Practice

	Ore, Gm/Au	Tailing, Gm/Au	Extraction, Per Cent	
Lab. test	13.4	0.9	92.8	1.0 (midds.)
Pilot mill	13.5	1.15	92.0	.....
Lab. test	9.9	0.8	86.8	5.4 (midds.)
Pilot mill	10.1	0.9	91.0	.....
Lab. test	6.5	0.4	92.2	2.5 (midds.)
Pilot test	5.0	0.6	88.4	.....
One-month average				
Pilot mill	5.0	0.75	85.3	.....
Cyanide plant	3.8	0.65	82.8	.....
Year average				
Cyanide plant	4.7	0.85	82.0	.....

course, was not taken into consideration in the calculations.

Even though recovery by flotation would only equal that of cyanidation, the advantages were many. Sufficient mill space was available to install flotation machines, and (with minor alterations) all other equipment could be made to conform with the new flowsheet. Expense for this installation would not exceed that necessary to get the additional equipment necessary to place the plant on a 600-ton cyanidation basis. Flotation would eliminate the sand plant. Daily slime-plant tonnage would be cut from 350 to about 20 tons.

The advantages of flotation in this case may be summarized as follows: power economy, labor economy, elimination of sand leaching, and a reduction of operating time in the slime department of 90 per cent. Simplification of the flowsheet is a major advantage; 90 per cent of the operating equipment is concentrated in one department, which makes a more compact and easier controlled plant. Flotation machines require only a small amount of repair or upkeep, so that they add little to plant maintenance cost. Inasmuch as more than half of the operating equipment would be scrapped, a proportionate decrease in operating delays from breakdowns and repairs could be expected,

with fewer repair parts in stock. In the treatment of heavier sulphide ores, such as are being developed in the South Group, flotation could be expected to give better recovery because of the more thorough treatment of the sulphide, in the form of a flotation concentrate, than could be expected if all the ore is to be cyanided. Another possible advantage is that at some future date, if increase in mill tonnage is desired (now being considered), this could be secured at the expense of additional grinding equipment only, necessary in either case, and more flotation machines. The expense of such an installation would be only a small fraction of that necessary for a similar tonnage in an all-cyanide flowsheet. Further, because of lower milling costs, flotation would permit the treatment of ores of still lower grade, thus adding to reserves.

Estimated flotation operating cost (only) was 0.28c. per ton, with a reduction in total milling cost of 0.50c. The economy is even more important than appears at first sight. Power is scarce; more than 100 kw. could be diverted to other purposes; 50 to 60 workers could be released that are urgently needed elsewhere.

Aside from lower operating and capital cost, flotation as an accessory treatment has solved difficulties of a metallurgical nature, such as removing minerals refractory to cyanide treatment, or removal of gold-bearing minerals amenable to cyanidation but contained in an ore carrying products deleterious to metallurgical treatment by cyanide. This case was peculiar in that we had an ore ideal for cyanidation, giving good recovery and containing no elements that would adversely affect cyanidation treatment, but its physical character was such that insurmountable problems arose in separating the dissolved gold from finely divided gangue.

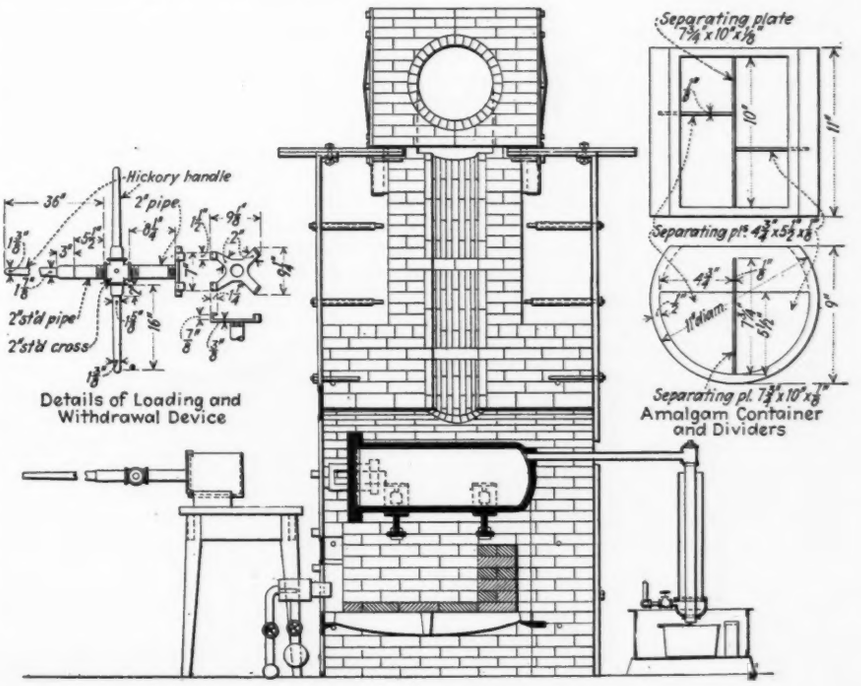
■

Handling Retorted Gold

A RED-HOT cast-iron container with its load of retorted gold is awkward to handle. At the refinery of the Homestake Mining Company, at Lead, S. D., a device is used that facilitates the removal of the container from the retort, as well as its charging thereto, whereby hazard of accident and discomfort to operators are reduced to a minimum.

The retort, shown in the accompanying line drawing, is oil fired. A removable table, with metal top, is used to facilitate charging of the containers, which are fitted with loose division plates that divide the retorted gold into pieces of a size convenient for charging subsequently into the melting crucible.

The device to handle the container is pictured in detail. It is made of a 2-in. standard cross piping, and other ordinary fittings, and is supplied with three hickory handles, with an arrangement of iron straps that permits easy attachment of the device to the container, previously pulled to the front of the retort chamber by an iron rod.



Employee Housing

Manitoba
Queensland
Chile

A. H. Hubbell
Associate Editor

CLIMATIC CONDITIONS at the Flin Flon mine, in Northern Manitoba, are relatively severe. Summers are hot and dry, the temperature from the middle of June to the end of August varying from 60 to 110 deg. F. in the shade, and averaging about 75 deg. The autumn is cool and frosty and is followed by a long, cold winter, lasting from mid-December until mid-April. During the latter period the temperature may remain at minus 30 deg. F. for weeks, dropping at times to 35 or 45 deg. below, or even to 50 deg. below, at night. For this reason, the Hudson Bay Mining & Smelting Company, which operates this property, has given careful attention to insulating its houses against the weather.

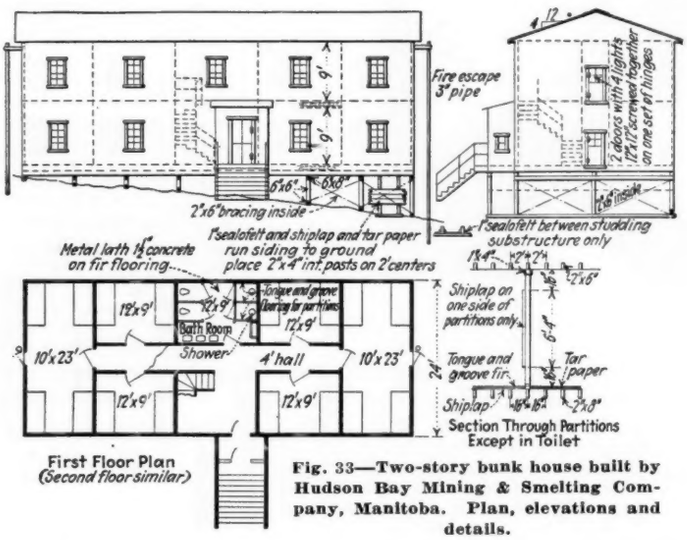

Fire escape 3" pipe
2"x6" bracing inside
1" sealofelt and shiplap and tar paper run siding to ground place 2"x4" int. posts on 2' centers
1" sealofelt between studding substructure only
Metal lath 1½" concrete on fir flooring
Bath Room
Shower
4' hall
First Floor Plan (Second floor similar)
Shiplap on one side of partitions only
Tongue and groove fir
Tar paper
Shiplap
Section Through Partitions Except in Toilet

Fig. 33—Two-story bunk house built by Hudson Bay Mining & Smelting Company, Manitoba. Plan, elevations and details.

Although one-third of the total area of this region is water, precipitation averages less than 10 in. for the entire year. Most of this occurs as rain, as the snowfall is light. In extremely cold weather the snow becomes like sand, adding to the difficulties of hauling.

To house its men during the construction of the plant and afterward, the company erected several two-story bunk houses. The floor plan is shown in Fig. 33, and an exterior view in another cut. The two floors are similar in layout, each having sixteen double bunks, a bunk house thus accommodating 62 to 64 men. These buildings were so designed that they can be and are actually being converted, as occasion requires, into apartment houses, each having four apartments for as many families. Likewise, they can readily be converted to their original state. The management, therefore, finds them satisfactory because of this flexibility. Single men only are housed in the unconverted bunk houses. No colored men are employed. One bunk house has been made somewhat better than the others, and is used for accommodating foremen, bosses, surveyor's helpers, and others of corresponding status.

A number of bungalows have also been provided. These are of two classes, differing only in size. They are shown in Figs. 34 and 35. The smaller (Fig. 34) serves to house the general foremen and shift bosses who are married. The larger ones, Fig. 35, house the general superintendent and various department heads. Besides these there is a staff house for the unmarried heads of departments and certain clerical employees and occasional visitors. This is not shown in the illustrations, nor is the boarding house, where the majority of single employees eat and which will seat about 600 men at once. Those assigned to the staff house eat in the dining room of that building, which will accommodate about 70 persons at a time.

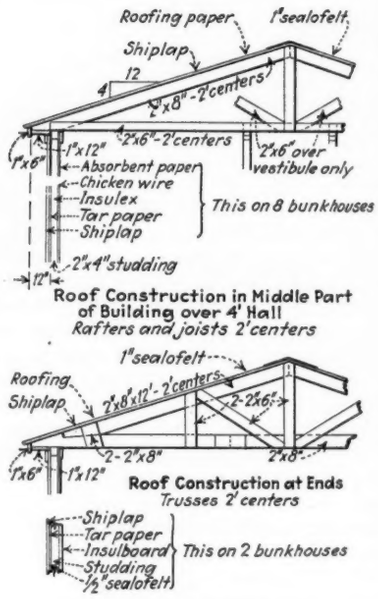


Fig. 33A — Details of bunk house roof and side-wall construction

These buildings are set directly on the ground. Where the site was rock, it was blasted level. All timber is British Columbia fir, and the siding British Columbia shiplap. In every building, the roof is of 1x12-in. British Columbia fir, covered with Johns-Manville roofing. Inside walls are of Tentest and various other materials.

In most of the bunk houses, 2x4-in. studding is used for the sides, with an inch of Celotex on the outside, followed by a good grade of roofing, and then shiplap. Because of the extremes of temperature to which the shiplap is subjected, the company had it made with a longer tongue than usual to allow for greater shrinkage and expansion.

One of the bunk houses of Hudson Bay Mining & Smelting Company. See Fig. 33. Note the pipe fire escape at the end.

Previous installments have dealt with employee housing in Northern Rhodesia; Newgulf, Texas; Matahambre, Cuba; Achotla and Monterrey, Mexico; Pecos Valley, New Mexico; Trona, California; Tyrone, New Mexico; and Clarkdale, Arizona.

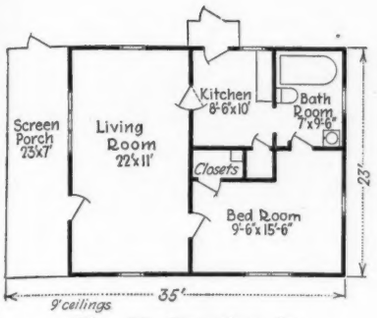
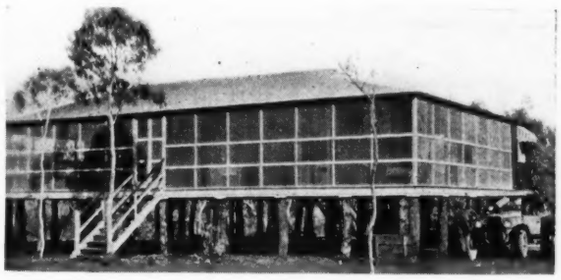


Fig. 34—Bungalow for married foremen and shift bosses, Hudson Bay Mining & Smelting Company. Floor plan.

Between the studding, Insulex, a gypsum product, was used for insulation. This is a powder containing an effervescent salt, which, when mixed with water, effervesces so as to form a porous material between the studding. It should be allowed to dry for at least two months before the inside is finished. In the construction of some of the buildings, ground cork and various other insulating materials were tried. Insulex, however, is by far the cheapest, according to the management, and when properly used has proved an excellent insulator against not only loss of heat but noise and vermin also. Other protection against vermin is afforded by fumigation. In general, this type of construction has been found cheap and satisfactory.

Side-wall construction of two of the bunk houses consists of Insulboard nailed on the inside of the studding, and



A typical Australian-type house, Mount Isa

on the outside a half inch of Sealofelt, followed by tar paper and shiplap.

Construction of partitions between rooms is indicated in detail in Fig. 33. For the roof, Celotex is laid on 1x12-in. B.C. fir board and covered with asbestos roofing.

Construction of the bungalows is in general like that of the bunk houses,

One of the few "California" type bungalows, Mount Isa

save that Johns-Manville white-top siding and roofing is used on the outside. This has been found satisfactory and, in addition, it makes an attractive house. R. E. Phelan, general manager, expresses the opinion that for a structure intended to last for years, ground cork, or possibly Zonolite, would be better as an insulating medium between the studding, if it could be had at reasonable cost; would last longer and would save money as compared with a cheaper insulation.

The respective costs of these buildings is as follows: Bunk house (Fig. 33) $8,009; three-room cottage (Fig. 34) $3,705, and the five-room bungalow (Fig. 35) $5,167. The staff house, which is not illustrated, cost $23,218.

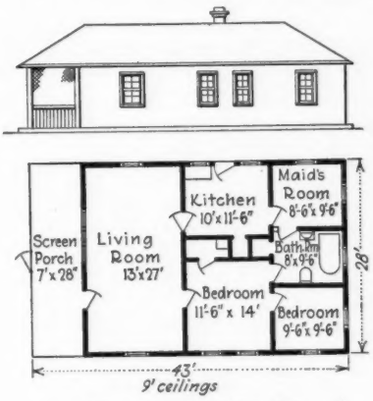
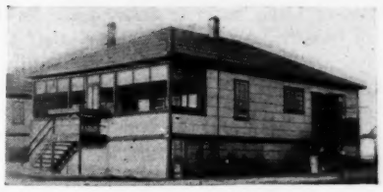


Fig. 35—Bungalow for general superintendent and department heads, Hudson Bay Mining & Smelting Company

Experience had thus far, according to Mr. Phelan, indicates that the best construction for bunk houses and cottages alike calls for 2x4-in. studding, with medium-grade tar paper roofing on the outside, followed by 1x12-in. planks covered with Johns-Manville white-top siding or Transite; on the inside of the studding chicken wire followed by absorbent paper, lath and plaster (or any boarding) in succession; and Insulex in the space between the studding. The inside lath and plaster finishing should not be applied until the Insulex has had a chance to dry for at least two months.

Running water under satisfactory pressure is provided in all dwellings, likewise electric light. In the bunk houses, one light per room is provided.



One of the large bungalows, Hudson Bay Company, shown in Fig. 35.

Occupants of the bungalows are provided with lights as they may elect.

A central steam-heating system supplies bunk houses and other dwellings. The management, however, is inclined to doubt whether this is convenient and satisfactory in so cold a climate. They hesitate to recommend it, owing to its relatively high cost for capital expense and for operation.

Toilets and showers are provided in the bunk houses, and ordinary bathrooms with the customary fittings in the bungalows.

Housing at Mount Isa Mines, Queensland, Australia—At Mount Isa the climate is much like that of parts of Arizona. The winters are mild, although a little heat in the houses is desirable at times. Summer temperatures frequently run as high as 110 or 115 deg. F. During the summer, the rainy season, which lasts from October to January, is at its height. Precipitation averaged approximately 12 in. annually for the five-year period 1926 to 1930 inclusive, having been as low as 5.04 in. in 1926 and as high as 27.06 in. in 1930, in which year floods were experienced. This rain comes principally in the form of heavy downpours. During a wet period, thunderstorms are frequent.

Only white, English-speaking employees are on the company's payroll, including both married and single men. The former live in single detached houses, and the latter in large dormitories. A large number of cottages of widely varying designs have thus far been built for the married men and their families. These include several dwellings that are quite like "California" bungalows for staff members and their families, a typical house of this kind being shown. It is carried on concrete piers.

Perhaps the dwellings most suited to the climate are the six or seven cot-

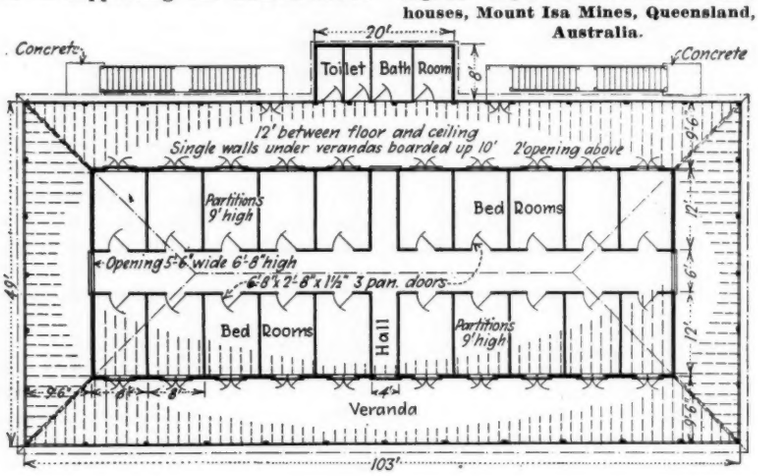
Some of the houses built for workmen, Mount Isa. Kitchen, bathroom, and lavatory only are on ground floor

tages of what may be termed the ordinary Australian type. These also are for staff members and their families. One of them is shown herewith. Its single story is raised above the ground, carried on posts 6 to 8 ft. high, permitting a free circulation of air underneath. Resembling these somewhat in having the living quarters raised on posts above the ground are the houses, several dozen in number, built for workmen, and having kitchen, bathroom, and lavatory on the ground and the bedrooms and large verandahs on the second floor. Kitchen, bathroom, and lavatory have concrete floors. One of these houses is here illustrated. The object in supporting the houses above the ground, either on posts or piers, is to make the living quarters cooler and to afford protection against the ravages of white ants, or termites, characteristic of the region. Once allowed to run unchecked, these will quickly destroy ordinary timber. Reports have it that they will not touch bloodwood, of which the posts are often made. Nevertheless, where wooden posts are used beneath the dwellings, the additional precaution is taken of drilling them and periodically impregnating them with an arsenic compound. The company also constructed a number of small temporary cottages for workmen. These were built with steel frames, corrugated-iron roofs and sides, and concrete floors, this for the purpose of trying out this type of construction. They proved to be a cheap type of construction. They were put up only to fill an emergency need during the construction period.

Two of the bunk houses, or "tradesmen's quarters," Mount Isa, while under construction. Note that the side walls of each floor reach only part of the way to the ceiling.

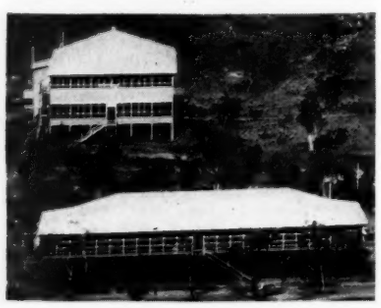
Housing for single men includes three two-story dormitories or bunk houses. A floor plan is shown in Fig. 36. Both floors are practically alike. Locally, these bunk houses are known as the "tradesmen's quarters," the term "tradesman" in Australia referring to all labor having a distinct trade as against unclassified labor. They have twenty rooms on each floor, which, with one man to a room, provide accommodations for 40 men per bunk house.

Fig. 36—Top floor of one of the bunk houses, Mount Isa Mines, Queensland, Australia.



Also for housing workmen is the structure, known as the mess hall, shown in Fig. 37. Twenty rooms on the upper floor of this building will accommodate 20 men. This floor is the same, in layout, as the floors of the bunk houses. The lower story is used as a dining room and is provided with eighteen large tables, as the drawing shows. Adjacent to this building at the back is the kitchen, a one-story building.

A building resembling the mess hall in general construction and appearance, but better finished, is the staff quarters shown in Fig. 38. The lower story accommodates a kitchen, dining room, smoking room, and four living rooms. On the second floor are eighteen living rooms, and in each corner of the verandah a bathroom has been provided. Both mess hall and staff house have 10-ft. screened-in verandahs on the second floor.



Mess hall (See Fig. 37), Mount Isa Mines, and, in the background, one of the bunk houses

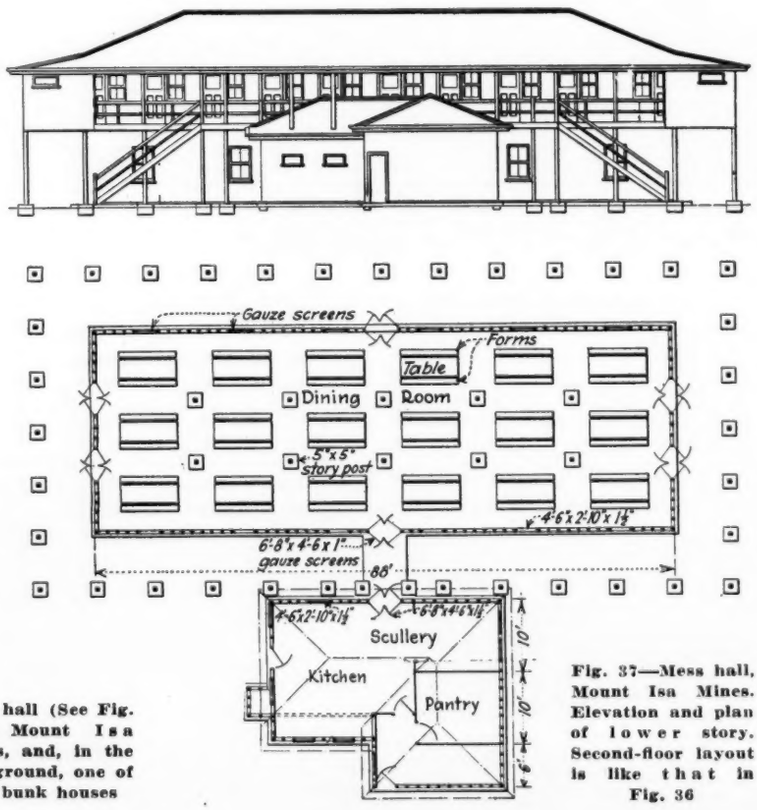


Fig. 37—Mess hall, Mount Isa Mines. Elevation and plan of lower story. Second-floor layout is like that in Fig. 36

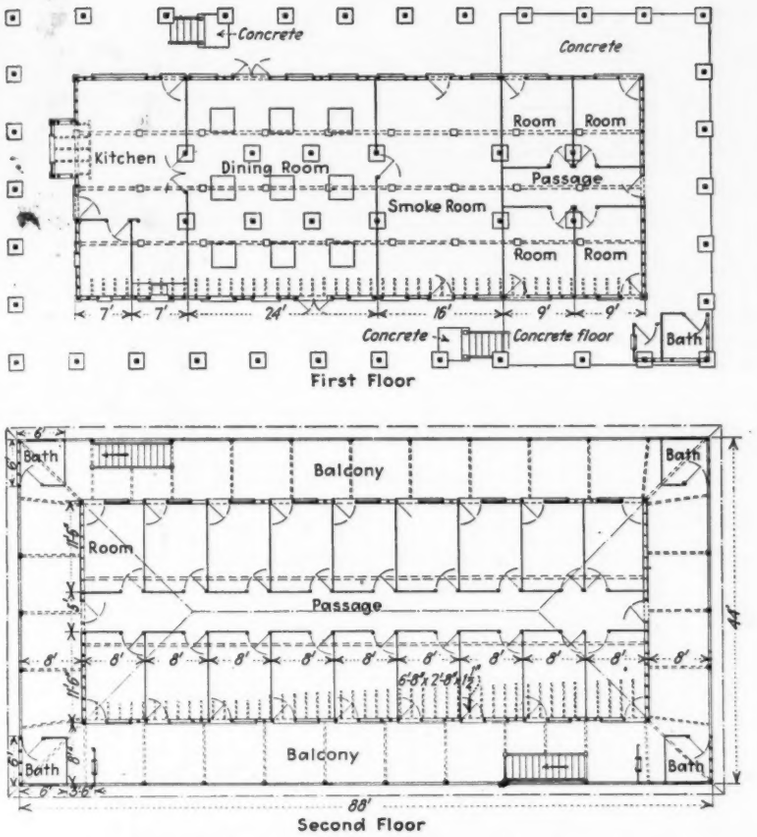


Fig. 38—Staff quarters, Mount Isa Mines. Plans of first and second floors.

The bunk houses, as shown in Fig. 36, are carried on posts, so that the lower floor is some distance from the ground. Each story has a screened verandah on all sides. The buildings shown in Figs. 37 and 38 are carried on small concrete piers, as illustrated.

Lumber used for construction is principally Oregon fir and some California redwood, imported from Seattle. The outside walls of the staff bungalows are of redwood shiplap, the insides being lined with tongue-and-groove boarding of Oregon fir. The houses of the Australian type referred to have Celotex walls and ceilings. Tongue-and-groove Oregon fir is used for walls in the mess hall and staff house (Fig. 37 and Fig. 38), and Compo-boarding in the tradesmen's quarters (Fig. 36).

Material almost universally used for roofing in this region is 20-gage galvanized corrugated iron. A few staff bungalows have roofs of Fibrolite, a corrugated asbestos-cement roofing. Corrugated iron's popularity as roofing is due to the fact that in hot weather it permits the dwellings to cool off quickly after sundown. Another reason, though one not applicable at the Mount Isa camp, which is now fortunately provided with an ample water supply, is that it serves satisfactorily to divert the rain water from the roof into galvanized-iron rainwater tanks, almost universally used throughout Australia.

The buildings described are provided with running water, electric light, and bath and toilet facilities. Rigid sanitation laws prescribe the use of individual septic tanks for each dwelling. The Mount Isa company has expended £300,881 on its community developments.

Housing of Chile Exploration Company, Chuquicamata—In Chuquicamata, Chile, situated in the Andes at 9,000 ft. elevation, the climate is usually mild, the temperature averaging about 68 deg. F. in summer, and about 60 deg. F. in winter, with extremes of about 25 deg F. at night in winter and 75 deg. F. during the day in summer. In consequence, little heat is required in the houses. Precipitation is too small to be measured, amounting to an occasional light rain or snowfall of short duration. At intervals of ten to fifteen years, heavy rains of short duration may be experienced. High winds are common, however, their velocity reaching as high as 70 miles per hour.

Earthquake shocks make wood framing more desirable than masonry or adobe construction. In consequence, reinforced concrete is used only to a limited extent for such purposes. The company has found concrete foundations advisable for houses. The foundations are without openings, in order to keep out not only the wind, but small animals, especially rats. Houses are without basements.

Douglas fir is most used for construction purposes, although a native wood, known as "lingue," is found satisfactory. No wood preservative is employed. Wood-destroying insects are unknown at this altitude.

Experience has shown wood-frame construction, with Douglas fir and Hyrib metal lath and cement stucco on the outside, to be best for general housing purposes. It is likewise thought reasonably fireproof and able to withstand the local climatic conditions. Sheet iron, or in some cases wood wainscoting, is used for lining the inside of workmen's houses. Floors may be of Douglas fir, although maple and oak are better. For this purpose the best native wood is lingue, but this must be thoroughly dry before laying, or shrinkage will quickly ruin the floor.

Corrugated sheet iron serves as roofing for the majority of the houses. Unless galvanized, it must be painted

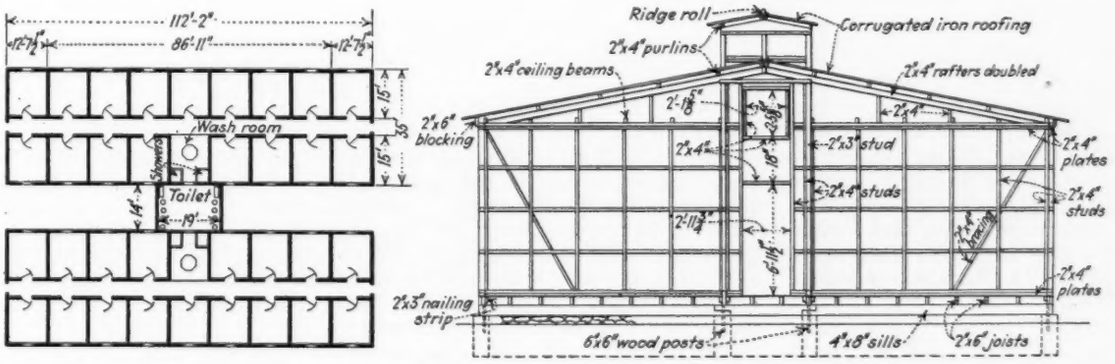


Fig. 39—Quarters for single workmen, Chile Exploration Company, Chuquicamata. Plans and end section of one wing.

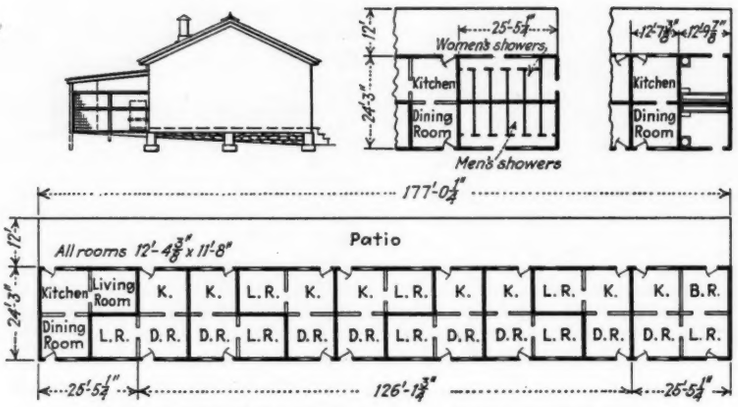


Fig. 40—Block of three-room houses built for married workmen at Chuquicamata. Plan and elevation.

frequently. In the construction of some of the houses copper has been used, starting sheets from the electrolytic cell house being employed.

Figs. 39 to 43 inclusive show various units of housing provided for the different classes of workmen, lower executives, and technical staff. All of them have concrete foundations, stucco outside, and are built with fir. Fig. 39 shows the quarters provided for single workmen, accommodating one man to six men per room, or eight in emergencies. Inside walls are of sheet-steel lining. Electric light, central water, toilets, and bath are provided. Ventilators in the roof are provided as shown in one of the details. Window sash are of cast iron, hinged to swing in. They are fitted with fixed screen poultry netting. Roofs are of corrugated iron. The cost to build in 1925 was $620 per room of 2,200 cu.ft., or 28c. per cubic foot.

Fig. 40 shows a block of three-room houses for married workmen. These have inside walls lined with sheet steel, but lack electric light, toilet, central water, and bath. Floors are of 1x4-in. tongue-and-groove lingue, except in the kitchens, where they are concrete. Each house or apartment contains a living room, dining room, and kitchen, and cost $1,020 in 1925. Each house containing 4,260 cu.ft., this cost reduces to 24c. per cubic foot.

Outside walls are of expanded metal and 1 in. of plaster. Room partitions are of 1x4-in. tongue-and-groove planking from floor to shelf, and of 26-gage sheet iron from the latter to the ceiling. Studs, girts, and plates are 2x3 in. for the most part. An exception is the kitchen, where partitions are of sheet iron. Ceilings are of 26-gage sheet iron and the roof is of 24-gage corrugated copper-bearing sheet iron. Windows are all double hung, 3x5 ft., and have small lights. Doors, used on the outside only, are 2 ft. 4 in. by 6 ft. 8 in.

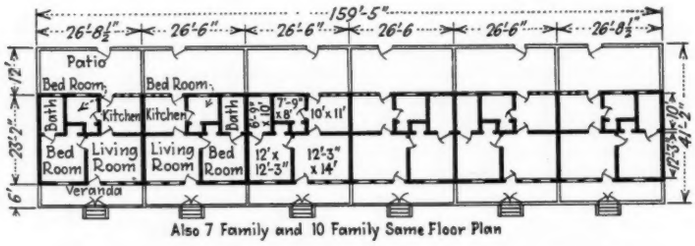


Fig. 41—Block of three-room houses built by Chile Exploration Company for married foremen. Plan.

Houses for married foremen are shown in Fig. 41. These have four rooms and bath each, with inside walls lined with fir. They are equipped with electric lights and heaters, running water, toilets, and bath. The cost of building, in 1925, was $2,720 per house of 6,200 cu.ft., or 44c. per cubic foot.

Fig. 42 shows the staff building for single men who are either foremen or on the technical staff. In construction and equipment it is the same as the dwelling shown in Fig. 41. Floors are 1x10-in. rough lumber laid on 2x6-in. joists. On top of this is laid a finish floor of edge-grain yellow pine, with a layer of deadening felt between. The building cost, in 1925, was $1,015 per room of 1,800 cu.ft., or 56c. per cubic foot.

The remaining type is shown in Fig. 43, and is intended for married men who are either foremen or on the technical staff. In construction, it is the same as the dwellings shown in Figs. 41 and 42. Construction cost in 1925 was $6,910 per house of 14,030 cu.ft., or 49c. per cubic foot.

In addition to the types of houses illustrated, the company has provided, for division heads and assistants, a type of dwelling having concrete foundations, and built with fir lumber, corrugated steel siding and roof, wood-board lining for the inside walls and ceilings, and equipped with electric light, heaters, a fireplace, water, toilets, and baths. Likewise, for department managers

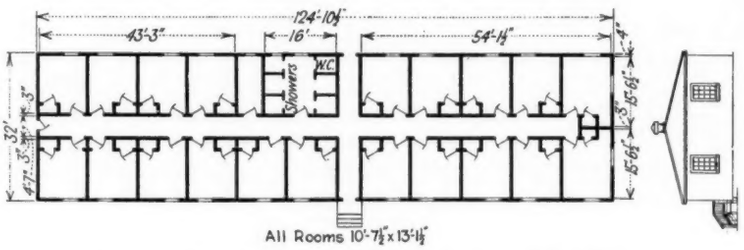


Fig. 42—Staff building at Chuquicamata, for single foremen and staff members. Plan and end elevation.

houses of similar construction and equipment are provided, but costing more. These two types of houses were built twelve to fourteen years ago. Duplicated today, the management states, they would probably be built like the various types illustrated. Their costs, estimated upon a basis of present-day costs of labor and material, are, respectively, $4,250 per house of 10,620 cu.ft., or 40c. per cubic foot, and, for the department manager's house, $8,450 per house of 21,220 cu.ft., or 40c. per cubic foot.

(To be continued)

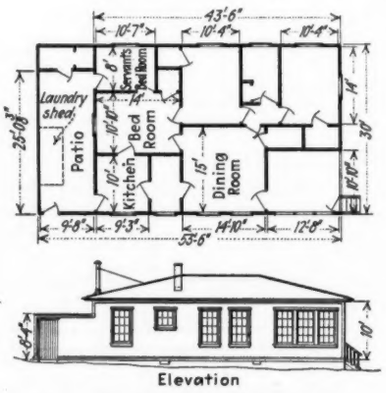


Fig. 43—Dwelling for married foremen and staff members. Plan and side elevation.

Cost of Cut-and-Fill Mining

Morris J. Elsing

FROM the point of view of production, the cut-and-fill method is the least important of the six primary mining methods. According to the U. S. Bureau of Mines, underground mines with an annual production of more than $100,000 produced nearly 100,000,000 tons of ore in 1929. Of this amount a little over 2,500,000 tons came from cut-and-fill stopes, or 2.6 per cent. The method was used wholly or in part at only 21 mines in the United States, of which two were gold, six copper, eleven lead-zinc, and two iron. Only a few mines employ this method solely, and records of the total cost of mining by the cut-and-fill method are therefore scarce. Published costs of most mining companies that use two or more methods do not usually give the complete details of the costs by each method. To tabulate itemized costs with the object of drawing definite conclusions is therefore difficult. Comparison is further complicated because of a lack of uniformity in the application of the cut-and-fill method. The three leading users produce 65 per cent of all ore mined by it, but in each of the mines the method is used in an entirely different way.

In 1929, the approximate production of the cut-and-fill stopes of the United Verde mine, Jerome, Ariz., was 700,000 tons. The Champion mine, Painsdale, Mich., was second, with 446,000 tons. Copper Range, Calumet, Mich., with its one-half interest in the Champion mine, produced about 540,000 tons of cut-and-fill ore. Calumet & Arizona ranks third, with about 400,000 tons of ore. Magma, at Superior, Ariz.; Pecos, New Mexico; and Block P mine, Hughesville, Mont., each produce about 100,000 tons annually from cut-and-fill stopes.

The method is used in mining large irregular orebodies such as at the United Verde and at the Calumet & Arizona, or in mining wide veins and lodes as at the Champion, Pecos, and Magma mines. It is also used effectively in mining very narrow veins that require the breaking of the wall rock in order to obtain a workable stoping width. This is the condition at the Block P mine, where the vein is from 1 to 4 ft. wide, and also at the molybdenum mine at Questa, N. M., where the vein is from 12 to 18 in. wide.

Although this is essentially a discussion of United States costs, Mexican costs (northern Sonora) are included because they refer to mines owned by United States companies, employing American methods and machinery, and supervised by American bosses. Many of the miners have worked in mines in the United States. Of these the Pilares mine, at Nacozari, Son., annually produces about 500,000 tons of ore from cut-and-fill stopes. Cananea is second, with 400,000 tons, and El Tigre third, with 60,000 tons. The American owned and operated Matahambre mine, in Cuba, produces 360,000 tons annually.

The ore deposits of United Verde occur as massive sulphide replacements of schist. They are of large horizontal extent but include smaller interfingering masses of ore. Chalcopyrite is the chief valuable constituent. The gangue consists of pyrite, quartz, and chlorite. Ore has been proved to a depth of 4,000 ft.

Generally, the massive sulphide is hard and dense, and stands well over large areas. The schist ore varies in strength and standing characteristics, permitting the use of the cut-and-fill method in some sections of the mine and necessitating square-setting in others. The porphyry ore can usually be mined by cut-and-fill. In 1929, 61 per cent of the production of the mine came from horizontal cut-and-fill stopes, and 3 per cent from incline cut-and-fill stopes.

Complete costs of mining at the United Verde appeared in Vol. 66 of the *Transactions* of the A.I.M.E. Cost of horizontal and incline cut-and-fill stopes, shown in Table I, are taken from this source. In general, incline stopes produce cheaper ore than horizontal stopes. Inasmuch as the ore mined in these incline stopes was exceptionally hard, the cost of breaking was high, resulting in a higher cost for the incline stopes than for the horizontal. An ore breaking cost more nearly representative of the average is shown in Table II (*Eng. & Min. Jour.*, May 29, 1926), and an average stoping labor cost is shown in Table III (*Min. Cong. Jour.*, April, 1930).

Table II—Ore Breaking Costs per Ton in Horizontal Cut-and-Fill Stopes, United Verde Mine

Miners	$0.170
Pluggers	0.070
Explosives	0.120
Compressed air	0.053
Drills	0.046
Steel	0.136
Total	$0.595
Tons per man-shift, total, underground	3.14
Powder, per ton, lb.	0.55
Timber, board feet per ton	6.95

Table III—Stoping Labor Costs per Ton, United Verdi Mine

	1929 Horizontal	1928 Incline
Miners	$0.12	$0.19
Plugger and car men	0.41	0.24
Waste men	0.07	0.06
Timber men	0.20	0.16
Miscellaneous	0.05	0.06
Totals	$0.85	$0.71
Tons per man-shift (stope), lb.	8.0	9.0
Explosives per ton	0.6	0.55
Timber, board feet per ton	6.95	5.90

The Champion mine, at Painesdale, Mich., is on the Baltic lode, which is an amygdaloidal bed dipping at 70 deg.

Table I—Stoping Costs per Ton, United Verde Mine

	Horizontal Cut-and-Fill Detail	Horizontal Cut-and-Fill Total	Incline Cut-and-Fill Detail	Incline Cut-and-Fill Total
Ore breaking, labor	$0.376		$0.786	
Explosives	0.140		0.257	
Compressed air	0.116		0.289	
Repairs	0.154		0.384	
Totals	$0.786	$0.786	$1.716	$1.716
Shoveling, labor	$0.440		$0.185	
Supplies	0.063		0.027	
Totals	$0.503	0.503	$0.212	0.212
Timbering, labor	$0.190		$0.153	
Timber	0.181		0.148	
Supplies	0.013		0.011	
Totals	$0.384	0.384	$0.312	0.312
Mining waste, labor	$0.017		$0.017	
Explosives	0.011		0.011	
Compressed air	0.009		0.009	
Repairs	0.011		0.011	
Totals	$0.048	0.048	$0.048	0.048
Delivering waste fill, labor	$0.252		$0.252	
Power	0.001		0.001	
Repairs	0.001		0.001	
Totals	$0.254	0.254	$0.254	0.254
Spreading waste in stopes	$0.274	0.274	$0.070	0.070
Total stoping	.....	$2.249	.....	$2.612
Recapitulation				
Total stoping, labor	.....	$1.549	.....	$1.463
Explosives	.....	0.151	.....	0.268
Timber	.....	0.181	.....	0.148
Compressed air	.....	0.125	.....	0.298
Electric power	.....	0.001	.....	0.001
Repairs	.....	0.166	.....	0.396
Supplies	.....	0.076	.....	0.038
Totals	.....	$2.249	.....	$2.612

This article is one of a series on the cost of metal mining in the United Statec.

Table IV—Michigan Cut-and-Fill Costs per Ton, 1912

	Baltic	Tri-mountain	Champion
Mining			
Superintendence and labor	$1.050	$0.961	$0.900
Powder and Supplies	0.166	0.175	0.153
Drills	0.077	0.143	0.081
Hoisting	0.076	0.088	0.052
Totals	$1.369	$1.367	$1.186
Tons hoisted	705,281	403,089	804,994

for a depth of 3,000 ft., then flattening slightly. Native copper occurs in this bed for a width of from 10 to 80 ft., the average stoping width being 17 ft. Both ore and wall rock are hard and stand well when openings are made. More than 40 per cent of the lode is composed of barren rock, which is sorted out underground. The horizontal cut-and-fill method has been used since 1905. At present a sublevel incline cut-and-fill method is being introduced successfully. Mining costs at the Champion are shown in Table XI. Costs of mining for the Baltic, Trimountain, and Champion for 1912 are shown in Table IV.

The orebody of the Campbell mine of Calumet & Arizona at Bisbee, Ariz., is a replacement deposit in limestone, 500 ft. long and from 50 to 250 ft. wide. The ore consists of chalcopyrite and bornite in a gangue of pyrite, silica and calcite. It is sufficiently hard to stand unsupported over considerable widths when properly arched and trimmed. Several modifications of the incline cut-and-fill method have been devised, which use one or two lead rows of square-sets. Direct costs of stoping from Information Circular 6289, U. S. Bureau of Mines, are shown in Tables V and XI. Note that Stope 68 was a double lead, inclined cut-and-fill, rafter-type bottom. Stope 90 was a single-lead, single-cap incline cut-and-fill stope.

The Campbell costs are somewhat higher than they would have been but for the fact that during this time many stopes were started or were in the initial stage of production.

Table V—Direct Stoping Costs, Campbell Mine, 1929 and 1930, Costs per Ton, Incline Cut-and-Fill

	Labor	Explosives	Timber	Air	Total	Tonnage
Stope 68	$0.620	$0.135	$0.155	$0.215	$1.125	51,970
Stope 90	0.700	0.100	0.130	0.200	1.130	20,894
Campbell mine	0.765	0.120	0.175	0.220	1.280	280,527
Lower Junction and Campbell	0.785	0.095	0.175	0.175	1.230	605,540

The ore at the Pecos mine, Tererro, N. M., consists of sphalerite, galena, chalcopyrite, and pyrite, with gold and silver associated with talc, mica, and chlorite schist. It occurs in irregular, often overlapping lenses in a shear zone in schist associated with diorite. Both ore and walls are generally loose. The ore zone dips at about 70 deg. The stoping width is 20 to 30 ft., but pinches down to much less. Ore has been developed for 2,000 ft. in length, and to a depth of 1,200 ft. At least 20 per cent of the stoping ground is sorted out in the stopes as waste filling. About 40 per cent of the production comes from cut-and-fill stopes. The method is used where the ore and wall rocks permit. Table VI, from Information Circular 6368, U. S. Bureau of Mines, shows comparative direct stoping costs, and Table XI the total cost.

Orebodies at the Magma mine, at Superior, Ariz., occur as shoots in an east-west fissure cutting Paleozoic sediments, diabase and porphyry. The dip of the vein is about 80 deg. In the lower levels the ore consists of bornite and chalcopyrite, with pyrite and some sphalerite. Crushed country rock and quartz form the gangue. The vein varies from 5 to 10 ft. wide in the narrower places, and from 20 to 30 ft. in the wider parts. The narrower parts are mined by rill square-sets. Wider parts are mined by alternate inclined cut-and-fill stopes, and pillars are later recovered by a modification of the Mitchell method. About 40 per cent of production comes from the cut-and-fill stopes.

Comparative direct costs of stoping by these methods are shown in Table VII, from I. C. 6168, U. S. Bureau of Mines; and the total cost of mining, reconstructed from these data in Table XI.

The orebody at the Colorado mine, Cananea, Son., Mexico, occurs as a pipe with the upper portion a ring of ore surrounding a waste core. This ring converges in depth and forms a solid orebody. The ore minerals are mainly chalcopyrite and bornite. Quartz is the principal gangue mineral. Maximum width of the orebody where it has the ring form is about 125 ft. The solid portion of the orebody is about 300x400 ft. The ore drills and breaks easily, and for the most part has good standing characteristics when openings are made.

During the first six months of 1929, 55 per cent of the tonnage produced came from cut-and-fill stopes. Of this amount 40 per cent came from incline stopes and 15 per cent from horizontal stopes. The remainder of the production came from development, shrinkage, and square-set stoping. Costs given in Table XII exclude mine-department expense, which includes superintendence, engineering, sampling, watchmen, time, and office. Comparative stoping costs are shown in Table IX, from I. C. 6247, U. S. Bureau of Mines.

Table VI—Comparative Direct Stoping Costs per Ton, 1929, Pecos Mine

Labor	Cut-and-Fill	Shrinkage and Fill	Square-Set and Fill	Underground Pillars	Average
Mining	$0.42	$0.34	$0.62	$0.62	$0.52
Mucking	0.44	0.12	0.57	0.49	0.49
Timbering	0.41	0.01	0.79	1.23	0.62
Filling	0.25	0.01	0.18	0.26	0.21
Total labor	$1.52	$0.48	$2.16	$2.60	$1.84
Supplies					
Mining	$0.09	$0.07	$0.12	$0.10	$0.10
Timbering	0.26	.....	0.40	0.42	0.33
Explosives	0.14	0.15	0.15	0.12	0.14
Total supplies	$0.49	$0.22	$0.67	$0.64	$0.57
Total	$2.01	$0.70	$2.83	$3.24	$2.41

Table VII—Comparative Direct Stoping Costs, per Ton, 1928, Magma Mine

	Incline Cut-and-Fill	Square-Set	Modified Mitchell
Labor			
Stoping	$0.320	$0.460	$0.260
Mucking	0.014	0.050	0.053
Timbering	0.400	0.530	0.380
Bonus	0.170	0.080	0.220
Total labor	$0.904	$1.120	$0.913
Supplies			
Waste fill	$0.030	$0.130	$0.230
Explosives	0.080	0.060	0.040
Timber, incidentals	0.340	0.480	0.280
Total supplies	$0.450	$0.670	$0.550
Total stoping	$1.354	$1.790	$1.463

Table VIII—Comparative Data, 1929, Pilares Mine

Method	Stoping per Ton	Bd. Ft. per Ton	Pounds Explosives per Ton	Tons per Man-shift
Horizontal cut-and-fill	$1.07	3.6	0.415	4.30
Inclined cut-and-fill	0.91	4.2	0.345	5.02
Square-set	1.62	13.6	0.325	3.36

Table IX—Comparative Stoping Costs per Ton, August-September, 1929, Colorado Mine

	Horizontal Cut-and-Fill	Incline Cut-and-Fill	Square-Set
Labor	$0.499	$0.495	$0.571
Explosives	0.131	0.140	0.074
Timber	0.252	0.282	0.629
Air, drills, incidentals	0.185	0.185	0.185
Filling	0.170	0.170	0.170
Totals	$1.237	$1.272	$1.629

The Iron Cap section of the Old Dominion vein at Globe, Ariz., varied from 3 to 40 ft. wide. Ore consisted of chalcopyrite and pyrite in a hard brecciated quartzite or shale between fairly good walls. Dip of the vein was 80 deg. With 50 men the average production per 8-hr. shift was 150 tons. One foot of development was required per 12 tons of ore mined. Ore was mined by the inclined cut-and-fill method, the average inclination of the filling being 34 deg.

Costs are given in Table XI. A miscellaneous item of $0.42 was equally distributed between stoping and miscellaneous mining. This team included supervision, engineering, assaying, pipe, nails, carbide, track, steel, and small supplies.

Production at the United Eastern mine, at Oatman, Ariz., started in 1917, and all known orebodies were exhausted in 1925. Two oreshoots were mined; one, the Tom Reed Extension, was 950 ft. long, 750 ft. high, and 48 ft. wide; the other, the Big Jim, was 850 ft. long, 450 ft. high, and 35 ft. wide. Total production was 732,000 tons, which yielded $14,500,000, mostly in gold.

The vein varied from a solid-quartz calcite filling to a series of parallel stringers separated by andesites. In the Big Jim mine the walls were more solid and this condition permitted more economical mining than was possible in the Tom Reed Extension. Average cost of mining for the Big Jim was $3.432 per ton, compared with $4.719 for the Tom Reed Extension. The horizontal cut-and-fill method was used almost exclusively. Costs are summarized in Table XI. A complete statement was published in Vol. 76 of the *Transactions* of the A.I.M.E.

At the Pilares mine, Sonora, Mexico, chalcopyrite with pyrite occurs as a breccia filling around an oval core 2,000 ft. long and 1,000 ft. wide. The area within the oval is a depression block of andesite and latite. The most persistant orebodies occur around the edges of the core, but irregular orebodies are also found within the central core. Both types of orebodies are irregular in size and shape. The ore is hard, and the waste walls are usually weak. When the ore is less than 30 ft. wide, stopes are laid out lengthwise. When the width is greater than 30 ft., stopes 30 ft. wide are laid out at right angles to the strike of the orebody. The maximum length of stopes is 70 ft. About 77 per cent of production in 1929 came from cut-and-fill stopes, of which 60 per cent were horizontal and 17 per cent inclined. About 18 per cent came from the square-setting of pillars, and 3.7 per cent from top-slicing. Some sorting is done in the stopes. The major amount of waste filling comes from surface glory holes.

Comparative data of horizontal cut-and-fill, inclined cut-and-fill, and square-setting are shown in Table VIII from I. C. 6307, U. S. Bureau of Mines, and the total cost of mining is shown in Table XII, I. C. 6247.

Table X—Mining Costs per Ton, Questa, N. M.

Period	1929	1930 (Ten Months)
Tonnage	7,846	10,112
Development	$7.48	$5.03
Mining	5.22	5.74
Total	$12.70	$10.77
Labor	$9.78	$8.34
Material	2.92	2.43
	$12.70	$10.77

Table XI—Cut-and-Fill Costs, per Ton, United States

	United Verde, Ariz. (Horizontal)	United Verde, Ariz. (Incline) A.I.M.E. Vol. 66	Champion, Mich., I.C. 6515	Campbell and Lower Junction, Bisbee, Ariz. I.C. 6289	Pecos, N. M. I.C. 6368	Magma, Ariz. I.C. 6168	Iron Cap, Globe, Ariz., A.I.M.E., Vol. 72	United Eastern, Oatman, Ariz. A.I.M.E. Vol. 76	Mine E, Michigan U.S. Bureau of Mines Bull. 306
Stoping									
Labor	$1.549	$1.463	$0.794	$0.785	$1.520	$0.904	$1.200	$1.236	
Supervision			0.056					0.060	
Explosives	0.151	0.268	0.103	0.095	0.140	0.080	0.240	0.279	
Timber	0.181	0.148	0.061	0.175	0.260	0.340	0.220	0.439	
Air, drills	0.125	0.298	0.128	0.175		0.280	0.420		
Power	0.001	0.001				0.134		0.087	
Miscellaneous	0.242	0.434	0.020		0.090	0.030	0.210	0.882	
Total stoping	$2.249	$2.612	$1.163	$1.230	$2.010	$1.768	$2.290	$2.983	$1.763
Development	0.670	0.670	0.490		1.218	0.736		0.501	0.598
Transportation	0.565	0.565	0.552		0.711	0.616	0.400	0.275	0.158
General	0.559	0.559	0.211		0.385	1.166		0.232	0.180
Miscellaneous	(a)	(a)	0.112		0.188	0.130	0.210	0.280	
Total mining	$4.043	$4.406	$2.528		$4.512	$4.416	$2.900	$4.271	$2.699
Tonnage	468,000	11,200	446,804	605,540	81,827	100,000	85,211	732,000	403,260
Year	1918	1918	1929	1929–30	1929	1928	1923	1917–25	1927

A.I.M.E., I.C., *E.&M.J.* and *M.C.J.*, refer to American Institute of Mining and Metallurgical Engineers, U. S. Bureau of Mines Information Circular, *Engineering & Mining Journal*, *Mining Congress Journal*.
(a) In addition to operating charges there was a charge of $1.243 per ton for taxes, injuries and damages, fire insurance, depreciation of mine plant, and incidental items.

Table XII—Cut-and-Fill Costs, per Ton, American Owned and Operated Foreign Mines

	Pilares, Son., Mexico, I. C. 6307	Colorado, Cananea, Mexico, I. C. 6247	Elisa, Cananea, Mexico, Personal	El Tigre, Son., Mexico, E. & M. J. 7/7/28	Matahambre, Cuba, I.C. 6145
Stoping					
Labor	$0.630	$0.375	.....	.....	$0.693
Supervision	0.181	0.132	.....	.....	0.095
Explosives	0.134	0.215	.....	.....	0.095
Timber	0.243	0.198	.....	.....	0.192
Air, drills	0.175	0.180	.....	.....	0.085
Power	.....	0.007	.....	.....	0.008
Miscellaneous	0.007	0.096	.....	.....	0.099
Total stoping	$1.370	$1.203	$1.990	.....	$1.26
Development	0.399	0.561	0.780	$1.81	0.711
Transportation	0.370	0.166	0.300	.....	0.388
General	0.250	0.278	0.480	.....	0.184
Miscellaneous	0.211	0.233	0.460	.....	0.019
Total mining	$2.600	$2.411	$4.010	$6.42	$2.569
Tonnage	643,000	142,006	96,472	80,570	364,746
Year	1929	1929 (Six Months)	1926	1927	1928

Table XIII—Statistical Data, Cut-and-Fill Method

Mine	Refer	Explosives Pounds per Ton Stope	Explosives Pounds per Ton Total	Timber Bd. Ft. per Ton Stope	Timber Bd. Ft. per Ton Total	Production Tons per Man-shift Stope	Production Tons per Man-shift Mine	Production Tons per Man-shift Total	Remarks
United Verde	A.I.M.E. Vol. 72	1.4		5.4		5.8			Horizontal, 1922
United Verde	A.I.M.E. Vol. 72	1.5		3.4		4.1			Inclined, 1922
United Verde	M.C.J. April, '30	0.60		7.0					Horizontal, 1929
United Verde	M.C.J. April, '30	0.55		6.0					Inclined, 1928
United Verde	I.C. 6440	0.55		7.0			2.8		1929
Champion, Michigan	A.I.M.E. Vol. 72		0.68		2.9		2.6	2.1	1921
Champion, Michigan	I.C. 6515	0.80	1.30	3.1	4.3	3.2	2.6	2.5	5 Months, 1930
Pecos, New Mexico	I.C. 6368		0.73		3.9 lin.			1.4	1929 all stopes
Pecos, New Mexico	I.C. 6368		0.85		2.8 lin.			1.7	1928 all stopes
Block P, Montana	I.C. 6416	2.57	2.85		4.8		1.5	1.4	1929
Block P, Montana	I.C. 6416				1.3 lin.				
Pilares, Mexico	E.&M.J. Oct. '27, '28	0.42		2.5		4.3	2.1		Horizontal 1927
Pilares, Mexico	E.&M.J. Oct. '27, '28	0.41		1.6		5.3			Incline, 1927
Pilares, Mexico	I.C. 6307	0.42		3.6		4.3			Horizontal, 1929
Pilares, Mexico	I.C. 6307	0.35		4.2		5.0			Incline, 1929
Cananea, Mexico	I.C. 6247		1.37		7.2		2.2	1.8	Half 1929
Matahambre, Cuba	I.C. 6145	0.44	1.02	0.80	1.1	2.8	1.9	1.8	1928
Matahambre, Cuba	I.C. 6145			0.13 piece	0.15 piece				

The ore at the Elisa mine, Cananea, Sonora, Mexico, was a hard garnet limestone containing chalcopyrite and a small amount of silver. Breaking costs were high because of the extreme hardness of the ore. About 40 per cent of the material broken in stopes was sorted out as waste filling. Because of the scattered and erratic character of the orebodies, excessive prospecting and development was necessary. With the exception of a small amount of ore taken from one or two square-set stopes in soft ground, the tonnage came from horizontal cut-and-fill stopes. The costs shown in Table XII are taken from personal notes.

The El Tigre mine, Sonora, Mexico, is 30 miles east of the Esqueda station on the Nacozari Railroad. The veins occur in rhyolite and rhyolite tuff and are almost vertical. Average width of the ore is only 1.7 ft., and the stoping width is 3.4 ft. which necessitates the breaking of the waste walls. This waste is used for filling in horizontal cut-and-fill stopes. A few inclined cut-and-fill stopes have been carried up and shrinkage was tried but found unsatisfactory. Costs are shown in Table XII.

The orebodies at the Matahambre mine, at Matahambre, Cuba, occur as large irregular pipes, usually of lenticular cross-sections of primary chalcopyrite associated with quartz and pyrite in well-defined fracture zones, usually between a thin bedded shale hanging wall and a quartzite footwall. The dip of the orebodies is from 42 to 45 deg.; length is usually several times the width. The hanging wall does not stand well, and the horizontal cut-and-fill method is used with cuts of small height. Ore is broken in stopes on contract at a price of from 60 to 70c. per car of 20 cu.ft., or about one ton. In some stopes, sorting is paid for at a price of 5c. more per car. Waste fill is spread in stopes by hand for 20c. per ton of fill and 5c. per ton with scrapers. About 30 per cent of the tonnage is mined by square-setting. This ore comes from the level or crown pillars just below and above the level. Costs are shown in Table XII.

Molybdenum Corporation of America operates a mine at Questa, N. M. The vein, in alaskite porphyry, is from 12 to 18 in. wide. Costs representative of the cost of mining very narrow veins are shown in Table X, from I. C. 6514, U. S. Bureau of Mines.

Conclusions—Although wide variation occurs in the costs of the cut-and-fill method, if they are segregated into two groups according to favorable and unfavorable ground conditions, a close similarity in each group is noticeable. Exceedingly hard ground, the necessity for close sorting, irregularity of orebodies, broken ground, and narrow veins necessitating breaking the wall rock add to the costs. Where favorable ground conditions exit, average stoping cost is approximately $1.25 per ton, the total mining cost being $2.50 per ton. With unfavorable ground conditions in the larger orebodies, the stoping cost is about $2.25 per ton, with a total cost of $4.25. Where the vein is narrow, necessitating the breaking of the wall rock, the cost depends largely on the width of the vein. For a 2-ft. vein the cost should be about $6 per ton, and for lesser widths, correspondingly higher.

In most instances the inclined cut-and-fill operations cost 10 to 15 per cent less than horizontal cut-and-fill. One exception, the cost given for United Verde, is explained by the fact that this particular orebody was much harder than the ore mined in the horizontal stopes. The most important saving permitted by the inclined method is in the avoidance of most of the shoveling of ore and waste filling. Most of the mines now using the horizontal method have tried the inclined method and have found it unsuitable because of the more rigid requirements for its successful application. Statistical data are shown in Table XIII.

■

Blasting Big Rounds Electrically

Frederick S. McNicholas

FOR the successful blasting of several machine-drill holes simultaneously by electrical means, the following procedure has been found essential at the Hidden Creek mine of the Granby Consolidated Mining, Smelting & Power Company, at Anyox, B. C.:

1. All conditions governing a big blast are studied and calculations for proper wiring and equipment are made from Ohm's Law. Standard resistance factors are also determined before final preparations are made.

2. Alternating current with a frequency of 60 cycles is used. Blasting should not be done with direct current from a trolley wire, inasmuch as ground leakage may be serious and only part of the charge may explode.

3. Existing power lines that carry current for lighting in drifts are not used, even though calculations show them to be of ample size to supply the current required. Worn insulation (especially around chutes) may allow current to leak to the ground through wet connections, causing only part of the round to explode. Instead, temporary lines of insulated wire of proper size are strung.

4. Where transformers must be used to secure the required current and voltage, the blasting switch is placed on the *secondary* leads from the transformer, and the current is maintained *on the transformer* prior to closing the blasting switch. In one blast, where the switch was placed on the primary leads running to the transformer, only part of the detonators exploded, although nothing was wrong with the wiring or connections. The switch was then placed on the secondary leads, and no more trouble was experienced.

5. Only enameled wire and waterproofed electric blasting caps are used, so as to avoid possibility of current leaking through wet or highly conductive ground. Cap wires are not cut off. To do so would alter the resistance of each series. The slight additional cost of enameled-wire caps is justifiable under almost any conditions.

6. For testing stope circuits and caps, only galvanometers with batteries supplied by the powder manufacturer are used. Use of other plant galvanometers and batteries is prohibited.

7. Blasting caps are connected in series, which in turn are connected in parallel to the main leads, with no branches and in the simplest manner. Each series must contain the same number of caps, even though additional caps be added. The number connected in each series is the minimum that conditions permit, because if one cap should fail, the series would fail. The governing factors are: the equipment available, the number of holes to be blasted, and the number of caps that may be exploded successfully in a series. The cap manufacturer recommends that not more than 50 caps be used in a series, although a series of one hundred has been exploded. Not more than fifty in a series are used.

8. Holes are loaded with the calculated amount of powder in a systematic manner, beginning preferably at the top of a stope and working down. When the loading crew has advanced far enough, the priming crew follows. The primer is carefully placed in the hole and pushed but not tamped into place. Two sticks of powder are placed on top of the primer and likewise pushed into place, but not tamped. No stemming is used. Care is taken not to scrap or jam the cap wires.

9. Following the priming crew come the electricians and helpers, who string heavy twine waist-high across the stope. Cap wires are scraped, tested, and connected in the required series. All connections are taped and fastened systematically to the heavy twine to keep them out of the wet. Electricians

Table I—Summation of Blasting Requirements for Varying Numbers of Holes Wired in Series

Caps in Series	Series Required	Amp. per Series	Total Amp. Required	Resistance One Series of Caps Ohms	Resistance of Total Series Ohms	Resistance of Wiring Ohms	Total Resistance of Circuit Ohms	Voltage Required	Transformer Connected to Give on Secondary Leads: 220 Volts — Available Safe Voltage 90% of 220 Volts	220 Volts — Available Safe Current 90% of 182 Amp.	440 Volts — Available Safe Volts	440 Volts — Available Safe Amp.
10	134	1.5	201	13.6	0.1	1.31	1.41	284	198	164	396	82
15	89	1.75	156	20.4	0.23	1.31	1.54	240	198	164	396	82
20	67	1.75	118	27.2	0.41	1.31	1.72	203	198	164	396	82
30	45	2.0	90	40.8	0.91	1.31	2.22	200	198	164	396	82
40	34	2.0	68	54.4	1.60	1.31	2.91	198	198	164	396	82
50	27	2.0	54	68.0	2.52	1.31	3.83	207	198	164	396	82

Table shows that the desirable connection of the transformer is to give 220 volts on the secondary leads; also that the round may be wired with either 30 or 40 caps in a series. Thirty caps in a series were used with perfect results.

then test each series and connect it to the main leads running out of the stope. These terminate outside the stope at the nearest point to the main lead-in wires. The connection to the main leads is the last to be made. The blasting transformer is then taken to the nearest substation and connected into the circuit. Power is generated at 2,200 volts, 3 phase, 60 cycles. A single transformer of suitable capacity is connected into the circuit, so that the blasting current is single phase, 60 cycles, of suitable voltage. The main lead wires are then strung from the transformer to the stope by tying with string to the trolley-hanger sprags. Test lamps of proper voltage are placed across the secondary leads from the transformer, and at the extremities of the main leads, to ascertain that proper voltage is available and that the current is turned on. After testing for voltage the blasting switch is opened, the connections from the main leads to the stope leads are made, and the blasting switch is closed, firing the shot.

Data

Resistance of electric caps:	Ohms
10-ft. wires	1.1024
16-ft. wires	1.3616
20-ft. wires	1.5344

Resistance of copper wire:	Ohms
1,000 ft. of No. 10 gage	0.9989
1,000 ft. of No. 8 gage	0.6282
1,000 ft. of No. 6 gage	0.3951
1,000 ft. of No. 4 gage	0.2485
1,000 ft. of No. 2 gage	0.1563
1,000 ft. of No. 1 gage	0.124

Current required per series of caps:	Amperes
10 caps or less in series	1.5
10 to 25 caps	1.75
25 to 50 caps	2.00

Holes to be blasted, 1,333
Cases of 50 per cent powder used, 436
No. 8, 16-ft. enameled-wire blasting caps used
No. 1 copper wire used, 3,000 ft.
No. 4 copper wire used, 1,000 ft.
No. 6 copper wire used, 1,740 ft.
One 40-kw. transformer available, 440 or 220 volts on secondary side
One 25-kw. transformer available 220 volts on secondary side

Calculation

Current Needed: This is calculated for several blasting operations, and the result summarized in Table I. The specific determination of the use of 30 caps in series is considered as follows:
Each series of 30 caps requires 2.0 amp., 1,333 holes divided by 30 gives 45 series total; 45 × 2.0 amp. gives 90 amp. required.

Voltage Needed: This is calculated for several operations and summarized: The specific instance of 30 caps is here taken.

	Ohms
Resistance 3,000 ft. No. 1 copper wire	0.377
Resistance 1,000 ft. No. 4 copper wire	0.242
Resistance 1,740 ft. No. 6 copper wire	0.689
Resistance of total wire	1.308

Resistance of 30 16-ft. caps, 30 × 1.3616 gives 40.8 ohms
Resistance of 45 series of 30 caps in each series, or 40.8 divided by 45, gives 0.91 ohms
Total resistance of circuit or 1.308 ohms plus 0.91 ohms, is 2.22 ohms. $V = IR = 90 \times 2.22 = 200$.

Power Available: The 40-kw. transformer was taken for blasting purposes. A factor of safety of 10 per cent is assumed as desirable. Wired for 220 volts on the secondary lead, the capacity of transformer is 40,000 watt divided by 220 volts gives 182 amps × 90 per cent = 164 amp. safe available current. 220 volts × 90 per cent gives 198 safe available voltage. Wired for 440 volts on the secondary leads, the capacity of the transformer is: 40,000 watt divided by 440 volts or 91 amp. This multiplied by 90 per cent gives 82 amp. safe available current. 440 volts multiplied by 90 per cent gives 396 volts safe available voltage.

Formulas and Calculations. The blasting current is single-phase alternating current and the load is entirely a resistance load, so Ohm's Law, $V = I R$, where V is voltage, I current in amperes, and R resistance in ohms, may be used for calculating all requirements. "The Blaster's Handbook" and pamphlets issued by powder companies give the resistance of electric caps and the current required to explode them, as well as standard resistance factors of copper wire. In calculating requirements, a safety factor of about 10 per cent should be allowed.

The data in Table II are based on operation over an eighteen-month period in which 23 large blasts, totaling 7,822 20-ft. holes, using 2,258 cases (112,900 lb.) of 50 per cent gelatin dynamite, were exploded. Total tonnage broken was 499,200.

The largest single blast was 1,335 20-ft. holes, using 436 cases (21,800 lb.) of 50 per cent powder, breaking 85,000 tons of ore.

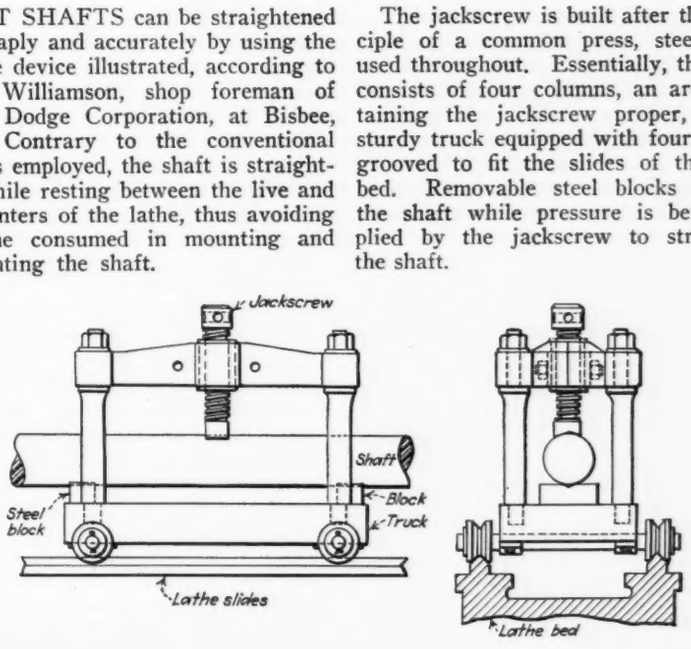
Table II—Blasting Data

Drilling	
Total machine shifts drilling	565
Total nipper shifts	310
Loading Holes and Carrying Powder	
Bar men, shifts	6
Miners, shifts	58
Muckers	42
Electricians, shifts	36
Data	
Total holes drilled	1,333
Total caps (No. 8 to 16-ft. wires)	1,380
Total cases 50 per cent powder	436
No. 1 copper wire used, ft.	3,000
No. 4 copper wire used, ft.	1,000
No. 6 copper wire used, ft.	1,740
Series of 30 caps in a series	46
Amperes required	92
Voltage required	202
One 40-kw. transformer connected to give on the secondary leads 220 volts	
Amperes available	182
Voltage available	220
Performance	
Holes per machine shift	2.4
Feet drilled per machine shift	48
Holes started at 3 in. Bottomed at 2 in.	
Holes per case of powder	3.6
Tonnage broken	85,000
Tons per machine shift	150
Tons per hole	64
Tons per case of powder	195
Pounds of powder per ton of ore	0.25

Portable Lathe Jackscrew

BENT SHAFTS can be straightened cheaply and accurately by using the portable device illustrated, according to J. P. Williamson, shop foreman of Phelps Dodge Corporation, at Bisbee, Ariz. Contrary to the conventional methods employed, the shaft is straightened while resting between the live and dead centers of the lathe, thus avoiding the time consumed in mounting and dismounting the shaft.

The jackscrew is built after the principle of a common press, steel being used throughout. Essentially, the press consists of four columns, an arch containing the jackscrew proper, and a sturdy truck equipped with four wheels grooved to fit the slides of the lathe bed. Removable steel blocks support the shaft while pressure is being applied by the jackscrew to straighten the shaft.



Homestake's Medical Service

THE Homestake Mining Company and its subsidiaries employed an average of 1,855 persons in 1930, a year of normal activity for the company. Of these 1,855 employees, 1,621 worked at Lead, S. D.; 133 were at Nemo, S. D., where the company maintains a timber camp and sawmill; and 61 were at Moskee, Wyo., where the company also conducts lumbering operations. In addition, 25 men were employed on a coal mine at Gillette, Wyo., and 15 workers were stationed at a hydro-electric plant at Spearfish, S. D. The company furnishes medical service without charge to all these employees and their dependents. At Lead the company maintains a 26-bed hospital, and a staff of five full-time physicians, who care for the company's personnel and their families in Lead and the immediate vicinity. Another physician resides at Nemo and looks after the employees and dependents there. The workers and their families at Spearfish, Gillette, and Moskee are cared for by local private practitioners, either on a part-time salary or on a fee basis. When there is need and the occasion permits, employees or their dependents at the outlying points go to Lead for hospital care. Approximately 25 per cent of the employees are foreign born and about 40 to 50 per cent are of foreign or mixed parentage. English and Welsh constitute the largest foreign groups, followed by Finns, Swedes, and Norwegians.

The company's medical department at Lead is housed in the Homestake Hospital, a three-story brick building on the main street of the town. The hospital has 26 beds and is well maintained and equipped. On the ground floor are the dispensary; a general waiting room; the physicians' offices; and the laboratory, physical therapy, X-ray, and record rooms. On the second floor of the hospital are patients' rooms, major and minor operating rooms, a sterilizing room, nurses' office, library, kitchen, and dining room. The third floor is occupied by the nurses' living quarters, patients' rooms, and linen and supply rooms. The hospital building was enlarged and completely remodeled in 1923. In 1929 the building alone was appraised (replacement value) at $101,535. Inclusion of the value of land, equipment, furniture, and supplies, increases the capital investment in the medical service to a total of $127,066.

Homestake service furnishes without charge all necessary medical and hospital care to the employees and their dependents. Although patients are admitted to the hospital whenever the physicians believe that hospitalization is needed, the facilities are used primarily for surgical cases: patients with contagious diseases are not admitted, and only those medical cases which cannot be satisfactorily cared for at home are hospitalized (for example, single men living in boarding houses are usually sent to the hospital when ill). Normal obstetrical cases are not ordinarily admitted unless the patient's residence is unsuitable for confinement or is at too great a distance from the hospital. The hospital will, however, accept normal obstetrical cases upon payment of $4 a day for hospitalization, if the patient agrees to engage a special private nurse for two days. Pathological confinements are hospitalized without charge. A company ambulance conveys sick or injured persons to the hospital. All required drugs and medicines, of whatever kind, are provided without charge.

The administration of the medical service is in the hands of the chief surgeon, who in turn is directly responsible to the general manager of the company. Because the service has been long established and because little change has occurred in the nature and extent of its work from year to year, the service has become, to a large extent, a self-contained, self-operating unit. Personnel of the medical department at Lead consists of five physicians, six registered nurses, and a dispensary nurse. The physicians are all employed on a full-time basis. The company does not permit them to engage in private practice, nor does the hospital admit private patients except, of course, in an emergency.

To all intents and purposes, the five staff physicians practice in much the same way as would an equal number of private practitioners. There is a marked tendency for each Homestake family to choose one doctor to go to or call for him only—in short, to regard him as the family physician. At least 90 per cent of the telephone requests for home calls specify which physician is desired. Each physician, therefore, has his own following; and, just as in private practice, some time must elapse before a new physician becomes firmly established—before he has as many patients as his colleagues. Patients have a free choice of physicians and may change practitioners at will, provided the transfer is made in an ethical manner. The free choice of physicians causes a certain amount of competition among the medical staff, since each desires to build up and to maintain a large practice. If one physician consistently lost patients to the others, he would soon feel in an uncomfortable position.

The total cost of maintaining the Homestake Medical Service in 1930 was $79,325.63, or $14.88 for each of the 5,332 individuals eligible to receive care. Table I presents a breakdown of this cost into its major items.

Of the total cost of the medical service, operating expenses constitute 85 per cent and fixed charges 15 per cent. A very large proportion of the fixed charges is made up of allowances for interest on the capital investment and depreciation on the hospital building and equipment. The largest single item of expenditure is that for the salaries of physicians. This item is equivalent to $5.39 per annum per individual eligible to the service, and constitutes 36 per cent of the total cost. The cost of providing hospital nursing service—that is, the salaries of the nurses—amounts to $1.41 per capita, or 9.5 per cent of the total annual cost. The cost of drugs, whether prescribed and obtained by patients from the local pharmacy or dispensed to patients in the hospital, aggregates $6,313.40, or 8 per cent of the total cost. This is equivalent to $1.18 per capita, a figure which confirms the conclusion reached in other studies of the committee; namely, that the cost of *necessary* drugs constitutes a very small part of the total cost of medical care. The expense for eyeglasses—borne by the patients and not by the company—amounts to $2,347.39 (3.0 per cent of the total), or 44c. per capita.

It should be borne in mind that the

For many centuries the practice of medicine has been carried on through an arrangement conventionally called private practice. Under this arrangement the patient receives medical service from an independent individual practitioner who is remunerated on a fee basis. Of late years, however, a tendency to question the efficacy and suitability of this traditional mode of medical practice has arisen. Accordingly, in order to appraise the advantages and disadvantages of other systems of rendering medical service, the Committee on the Costs of Medical Care adopted as one of the main lines of its program of study the investigation of "organized facilities" providing medical care. The following study falls within this field.

Abstract from "The Medical Service of the Homestake Mining Company," by Louis S. Reed, Ph. D.; Publication No. 18, Committee on the Costs of Medical Care; August, 1932.

$14.88 which represents the per-capita cost of the Homestake medical service is not the total per-capita cost of medical care for the eligible population. The company employees and their families may, in addition to the service furnished them without charge, obtain dental care at their own expense, and must pay individually for such nursing care in the home as they require. Some Homestake families occasionally consult private practitioners. Others doubtless patronize the chiropractor in Lead or the one in Deadwood, four miles away, and all perhaps buy patent medicines. How much is spent by Homestake people in this way is not known.

Not all of the costs of maintaining the medical department should be classified as costs of caring for sickness. A part of the expense must be charged to medical service for industrial injury cases, an expense which under the compensation law the company in any event would have to bear.

Of 436 admissions to the hospital in 1930, 61 were industrial accident cases. These cases required 1,026, or 24 per cent, of the 4,278 hospital patient-days of care given during the year. On this basis $8,947.26 of the $37,280.24 which it cost to maintain the in-patient service should be credited against the cost of treatment for industrial injuries. No records exist by which the cost of out-patient and home, and laboratory service can be accurately divided between care for industrial injuries and care for sickness and non-industrial injuries, but care for industrial injuries probably constitutes about 4 per cent of the total cost of providing these services. Thus, approximately $10,630 (13.4 per cent of the total cost, or $1.99 per capita) can be subtracted as a charge for industrial medical care as distinguished from non-industrial care for workers and their dependents. The total cost for care in non-industrial sickness becomes $12.89 per person eligible to receive care.

The annual cost of the Homestake medical service is $14.88 per person of the eligible population. What volume of medical service is provided for this sum? Records of the services performed make it possible to answer this question. During 1930 the physicians of the staff made 11,047 home calls, and attended 136 confinements, of which 29 were in the hospital and 107 at the patients' homes. No complete count of the number of office visits is available. However, the dispensary nurse counted the number of visits in four different weekly periods during the year, and on this basis estimated that during 1930 there were, after eliminating all duplications, 46,220 dispensary and office visits. Some of these visits were for trivial ailments or were made under such conditions that a private practitioner would ordinarily charge no fee; for instance, the number of dispensary and office calls mentioned in the foregoing included visits made simply to obtain authorization for the refill of a prescription. In addition to the above services, 1,233 physical therapy treatments, 1,033 employment examinations, and 511 eye refractions were performed, as well as all necessary laboratory and X-ray work. Finally, all necessary professional services were given to 88 medical and 348 surgical cases cared for in the hospital. On the average, there were 2.1 home calls, 8.7 office and dispensary visits, and 0.8 of a day of hospital care for each person in the eligible population.

Does the experience of the Homestake medical service indicate that economies may be achieved if medical service is rendered by an organized group rather than by individual practitioners? The evidence on this point is presented in Table II, which lists the services performed by the Homestake medical staff, together with an estimate of the probable cost of these services if they had been purchased at the fees generally charged by practitioners in the Lead district.

Conclusions—The principal facts and conclusions of this study may be summarized as follows:

1. The Homestake Medical Service, consisting of 6 full-time physicians and 1 part-time physician, 5 of whom comprise the staff of a 26-bed hospital owned and operated by the company, gives complete medical and hospital care, except dentistry, to a total of 5,400 persons—1,800 employees and their dependents.

2. The cost of this service in 1930 was $79,325.63, or $14.88 per person in the eligible population. This includes fixed charges on plant and equipment. If there is subtracted from this amount the estimated cost of providing care for industrial injuries—an expense which the workmen's compensation law compels the company to bear—the per-capita cost becomes $12.89. If the service were to be entirely self-supporting, a contribution of $3.24 (exclusive of the costs of service for industrial accidents) would be required each month from each employee.

3. The entire cost of the medical service, with the exception of the cost of eyeglasses, is borne by the company. However, since the company does not regard its medical service as philanthropic or charitable work but maintains the organization in order to improve the health and efficiency of the employees, to reduce labor turnover, and to strengthen morale, and since workers undoubtedly take the factor of "free" medical service into account when accepting employment, the provision of medical service may properly be regarded not as a free gift, but as a form of wages—wages "in kind." It would be a mere bookkeeping device to

Table I—Cost of Homestake Medical Service

Expense Items	Cost	Per-Capita Cost	Per Cent of Total Cost
Operating Costs			
Personal services:			
Salaries of physicians	$28,702.00	$ 5.39	36.2
Salaries of nurses	7,496.09	1.41	9.5
Wages of servants, janitor, etc.	3,352.05	0.63	4.2
Non-personal services (electricity, telephone, water, laundry)	4,221.05	0.79	5.3
Non-medical supplies (food, fuel, household and miscellaneous supplies)	7,848.32	1.47	9.9
Medical, surgical, and X-ray supplies (not including drugs)	1,992.40	0.37	2.5
Repairs and replacements	2,150.71	0.40	2.7
Expense for ambulance, physicians' transportation, etc.	2,304.45	0.43	2.9
Drugs	312.72	0.06	0.4
Allotment for general administration, bookkeeping, and legal services	400.00	0.08	0.5
Supplies for patients:			
Prescriptions	6,000.68	1.12	7.5
Eye-glasses	2,347.39	0.44	3.0
Total operating costs	$67,127.86	$12.59	84.6
Fixed Charges			
Interest(a)	6,187.09	1.17	7.9
Depreciation(b)	4,597.89	0.86	5.8
Taxes and insurance	1,412.79	0.26	1.7
Total fixed charges	$12,197.77	$ 2.29	15.4
Grand total	$79,325.63(c)	$14.88	100.0

(a) Five per cent of the total value of hospital land, building, and equipment.
(b) Calculated at 3 per cent on building, 5 per cent on heating plant, call system, and lighting fixtures, and 10 per cent on furniture and medical equipment.
(c) The net cost of the medical service to the company is $75,304.77. The cost of eye-glasses is borne by the patients, and the hospital has a small amount of income from the fees of normal confinement cases and from non-company emergency cases.

Table II—Value of Homestake Medical Services

Service	Number Performed	Value Per Unit, Based on Prevailing Fees in Private Practice in Adjacent District	Value
Home calls	11,047	$ 3	$ 33,141
Dispensary and office visits	46,220(a)	2	92,440
Confinements (at patients' homes)	107	35	3,745
Laboratory tests	2,614(b)	1–5	3,253
Physical therapy treatments	1,233	1	1,233
X-ray:			
Treatments	75	5	375
Gastro-intestinal examinations	105	25	2,625
Barium enemas	15	5	75
Pictures	515	5	2,575
Refractions and eye-glasses fitted	511	10	5,110
Employment examinations	1,033	1	1,033
Professional services to medical cases in hospital	88	10(c)	880
Professional services to surgical cases in hospital	348	40(c)	13,920
Days of hospital care	4,278	3.50	14,973
Total			$175,378

(a) After eliminating calls made to obtain special services charged for under other headings.
(b) After eliminating laboratory tests routinely performed for hospital patients. These are ordinarily included in a hospital charge.
(c) It is difficult to determine the appropriate fee for medical and surgical cases when the severity of the cases in question is not known. Accordingly, a figure of $10 for each medical case and $40 for each surgical case has been assumed. These are minimum figures, as will be seen by an inspection of the list of cases treated

credit each employee with an additional $3.24 a month, and to deduct this amount from his wages to pay for the maintenance of the medical department.

4. A comparison of the amount and cost of medical services received by the Homestake population with charges for, and volume of, medical service received by families of comparable income who obtain medical care in the ordinary manner, indicates (1) that the cost of service for the Homestake population is approximately the same as the expense which families of an equivalent income ordinarily incur for medical care, and (2) that the Homestake population receives more office and home calls per capita than the outside population, and about the same amount of hospital care.

5. When the amount of service performed by the Homestake doctors is priced at the prevailing fees of local private physicians, it appears that the total financial value of the services rendered is substantially in excess of the cost of maintaining the company medical service.

6. If the Homestake population at Lead were cared for by five private practitioners instead of by the five Homestake physicians, the private practitioners would probably have net incomes equivalent to the salaries of the Homestake physicians. But under private practice each unit of service would cost more, and the population would receive less medical service than under the present arrangement.

7. The medical service is, on the whole, considered satisfactory by its patients, although some individuals utilize outside facilities at their own expense. The company medical staff and the hospital are well regarded by the people of the community.

8. The incomes of the Homestake physicians compare favorably with those of private practitioners in South Dakota and adjacent states.

9. On the whole, the physicians of the service believe that their type of practice possesses advantages over private practice both from the patient's and the physician's point of view. Chief among these advantages is the fact that patients receive medical attention earlier in sickness and irrespective of their ability to pay. Other advantages are the stability of the physicians' income, the regular hours, the annual vacation with pay, and the opportunity for more frequent consultations.

10. Although the Homestake physicians are sometimes called unnecessarily, they do not believe that they are imposed upon or that the abuse of the privilege of medical service without specific charge is significant.

11. Without exception, all the physicians were most happy to be released from the necessity of making and collecting charges, and from similar business details incidental to private practice. They found satisfaction in their ability to give patients all the attention necessary—to call as often as need be, to hospitalize patients, or to prescribe laboratory tests — without considering the financial status of the patient under treatment.

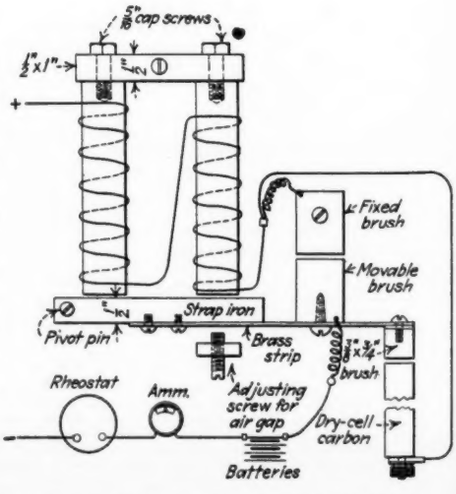
12. In the opinion of the physicians concerned, a direct financial relation between physician and patient—the payment of a fee for each service—is neither essential to the performance of service of good quality nor to the maintenance of a close and continuing personal relationship.

■

Automatic Reverse Current Cutout

AN AUTOMATIC CURRENT cutout for battery charging devised by Thomas C. Hulton, master mechanic, Cia. Minera Chambas, Abras Grandes, Cuba, was described in a recent issue. Mr. Hulton has made further improvements by adding a pair of carbons from a dry cell, which together offer sufficient resistance to prevent discharging of the batteries when the current is off, at the same time allowing the actuating magnet to be energized when the 110-volt charging current is again resumed. When the magnet armature moves up, the charging circuit is closed, and the small brush-carbon circuit is opened. The adjusting screw is used to maintain the proper opening between the two sets of carbons.

The improved cutout, showed in the accompanying sketch, has a magnet of 1-in. round stock, wound with 120 turns of No. 20 annunciator wire. Contact is made by a pair of old $\frac{5}{8}$x1$\frac{1}{4}$-in. generator brushes with pigtails, the movable brush being fastened by a strip of springy sheet brass to a $\frac{1}{2}$x1-in. iron strap secured by a pivot pin. An interruption in the power causes the movable brush to drop down immediately, thus breaking the circuit. The parts are so adjusted that the $\frac{3}{8}$x$\frac{3}{4}$-in. brush falls into contact with the telephone carbon when the current fails or the circuit opens for any other cause.



Chemical Activity the Cause of Flotation?

Otis D. Welsch

ONE is impressed, in searching literature on the reason for flotation, to find such marked similarity of opinion among investigators. Ernest Gayford[1] summarizes the general view as follows: "The two prime requisites to flotation are (1) the formation of bubbles; and (2) creation of selective preference of the bubbles for metallic particles instead of for gangue particles." I suggest study of the reason for flotation from another angle: on the flotation of mineral particles, sulphide or non-sulphide, as the result of buoyancy created by the development at the surface of the particle of a volatile substance. The majority of investigators seem to be fairly well agreed on what happens in the final step in flotation; as to the function of the bubbles introduced mechanically into the pulp; that for these bubbles to function most efficiently they must be minute. These mechanically made bubbles are pictured as attached to mineral particles, thereby imparting buoyancy to them and causing them to rise to the surface of the pulp, forming a mineralized froth consisting primarily of bubbles coated with the floated mineral. One easily comprehends how a mineral particle becomes buoyant if at its surface appears, or is attached, a bubble of gas or air. To comprehend how the bubble becomes so intimately associated with the ore particles is not so simple.

The generally accepted idea of the mechanism of flotation assumes that the surface of the floated mineral particle is dry or of a water-repellent nature at the instant of attachment to the introduced bubble. For the non-floated mineral, the assumption is that the surface has been wetted, and, consequently, that no attachment is possible between bubble and mineral. Present-day views on the subject differ little from those held several decades ago, although the explanation may be more complicated and consequently more difficult for the average millman and metallurgist to comprehend. Nevertheless, although the reason is in doubt, the art has progressed so that most minerals of economic importance can be concentrated by the flotation process.

A. S. Adams[2], in outlining conditions required for flotation, states: "It is fairly clear that in order to accomplish the phenomenon of flotation, we must have matter present in each of the three phases; i.e., a solid particle suspended in a liquid comes in contact with and adheres to the surface of a gaseous bubble. To achieve this minimum requirement for flotation, we must have absolutely pure matter in each of the three phases, and the bubble and solid must always have exactly the same size and shape." In other words, bubbles and mineral make up the froth in modern practice; bubbles are buoyant; they are of specific gravity less than that of water; they are present in the flotation pulp. Consequently, one may assume that these bubbles function as the agency that furnishes the transportation to carry the mineral particle to the surface.

One purpose of this paper is to suggest the probability that a mineral particle obtains transportation to the surface primarily by means of bubbles resulting from chemical reaction at its surfaces. This chemical activity, when culminating in flotation, results in the production of volatile matter. If one admits that chemical activity between the surface of the mineral particle and the soluble constituents (added as flotation reagents or inherent in the water of the ore pulp) may result in an evolution of gas, then the source of the floated mineral particle's buoyancy can be more definitely established and possibly proved; a theory can be advanced that will explain flotation in all its applications. This gas, as its volume increases —by decrease in temperature, pressure, or by continuation of the chemical reaction—tends to form a bubble at the position of least hydrostatic head, or at the point nearest the surface of the pulp. As the chemical action continues, or the pressure is decreased, the bubble grows in size, thereby increasing buoyancy or, rather, decreasing the specific gravity of the combination of mineral particle and volatile substance. When the specific gravity becomes less than that of the liquid, the mineral particle can pay for its transportation, and eventually arrive at the surface of the pulp. This rising to the surface is aided, in an efficient flotation apparatus, by proper circulation and mechanically introduced bubbles, which, by their bouncing action, can assist materially. Proper circulation is given an important rating in modern practice. It should be. Actual flotation of the mineral is accomplished at a comparatively shallow depth—from a point near the surface of the pulp. Proper circulation places the mineral particle in this position.

Flotation in the same ore pulp of one mineral, such as galena, in preference to other minerals, such as pyrite or quartz, indicates a selective chemical reaction involving the galena and the soluble constituent of the collecting reagent employed. The chemical reaction occurs at the surface of the galena, and the result is the production of one product at least that is volatile and insoluble in the liquid employed. Chemical reaction, if any, at the surface of the non-floating mineral results in the production of soluble or insoluble substances, neither of which by itself is capable of promoting buoyancy. So that this developed volatile substance may function as an agent that will enhance the buoyancy of the mineral particle, no reaction should occur between the volatile substance and the other constituents of the ore pulp.

R. S. Dean[3] stresses the foregoing principle when he gives briefly his conception of the Adams gas-adsorption theory, as follows: "The point of the gas adsorption theory, as I conceive it, is merely that the surface of the mineral must be completely covered with gas, being therefore a weighted bubble. What the gas is, so long as it does not react with any of the constituents of the pulp, makes little difference. Many gases, however, react with constituents of the pulp." Dean apparently recognized the possibility of covering the mineral particle with a gaseous envelope. But whence this gaseous covering? Can it materialize as the result of chemical activity or is it the result of a physical phenomenon designated as adsorption?

A. W. Allen[4] offered valuable suggestions relative to the possibility that chemical reaction or "chemical affinity" exists between the surface of the mineral particle and the "air used as a lifting agent," or of the gas, other than air, that promotes flotation. He recognized chemical activity at the surface of the mineral particle, but he did not appreciate the possibility that chemical activity might develop the bubble that would remain attached to the mineral particle and give it buoyancy. He also recognized a factor that is believed to be of prime importance if a "gas development at the mineral surface" is to be accepted; that is, the condition of the surface of the mineral. For example, if galena is to be floated, the average investigator in the past considered the surface of the mineral particle to be lead sulphide; but one must consider chemical activity (before and throughout the grinding operation) at the surface of the mineral particle. Such chemical activity alters the surface of the mineral particle, and no longer is lead sulphide the outer covering of the original particle. The literature offers considerable evidence in support of this conclusion. Allen states[5]

that: "The wide variation in results with and without the desorption treatment suggests to me that we have been prone to consider the composition of a mineral substance according to chemical analysis, forgetful of the fact that the condition of the surface and the character of what is adsorbed thereon constitute a powerful factor on the results of any treatment involving delicate or initial reactions. The degree to which a mineral is ground must also be taken into consideration. Surface activity increases in direct ratio to the area of surface exposed." In other words, have too many investigators based their conception of the theory of flotation on the chemical composition of the mineral particle as a whole, rather than on the possible chemical composition of that portion of the mineral exposed to atmospheric agencies or other sources of chemical activity?

A. F. Taggert and others[8] present evidence to show that mineral particles when exposed to oxidizing conditions, in crushing and grinding circuits and even the flotation circuit, become filmed or have their surfaces changed to an oxidized condition, as illustrated, in the case of galena, by a "form of lead sulphate." To accept this idea of surface alteration of the mineral particle should not be difficult. Minerals are susceptible to change by chemical agencies. They are subject to conditions capable of producing alteration at exposed surfaces from the time the ore is mined until the flotation apparatus has effected its purpose.

Ore exposed to agencies capable of promoting chemical activity offers evidence as to the nature of surface alteration of mineral particles. In the upper zone of such an orebody the oxidation has usually progressed to such an extent that little or no primary mineral remains. In what was once galena, the predominant mineral is the lead carbonate; in chalcopyrite, the resulting product may ultimately be copper carbonate. In the change from galena to cerussite, the sequence of change is first to the sulphate and then to the carbonate. If this change is possible in nature, is not the same sequence of change a matter of possibility, at least, under the conditions prevalent during crushing, grinding, and flotation operations?

Taggert and others base their theory primarily on the assumption of a water-repellent surface produced by the reaction between the soluble constituent of the collecting reagent and the surface film of the mineral particle, which in galena is considered as a "form of lead sulphate." Can this conclusion be considered as final if the change produced by nature in her slow, methodical way is a parallel sequence of alteration? To correlate known methods of flotation should be possible; for instance, the flotation of galena and the flotation of cerussite with ethyl xanthate. In the former case only a small amount of reagent is necessary; in the latter, a rather excessive amount. The literature has been searched, but no reference found of floating lead sulphate with an excess of ethyl xanthate.

If this argument has merit, then, to study the mechanism of the flotation of mineral particles, knowledge is required of the condition of the mineral surface that is prepared or in the proper condition to become buoyant, under conditions identical with those obtaining in actual practice.

Volatile matter, if properly captured, can impart buoyancy to solid matter. An ore pulp consisting of finely ground ore, water, oil, and chemicals can be subjected to a process, consisting of agitation, aëration, and circulation, that will result in the selective buoyancy or flotation of one or more of the contained minerals. To float in the ordinary sense of the word, such minerals must have acquired buoyancy. Buoyancy of solid matter in a flotation pulp must be a function of acquired volatile matter, whether by adsorption of gas (physical) or air; chemical affinity for gas or air; attachment to gas or air bubbles, or to the development or expansion of gas at the mineral surface.

K. A. Mickle's investigation of the flotation of minerals several years ago[9] should throw light on the reason for

flotation, if considered from the standpoint of the condition of the mineral surface and the possibility of chemical activity involving the mineral surface in the production of volatile matter. As suggested by C. E. Locke[9]: "It would be a wise thing for us to go back and see what Mickle did." A few observations from Mickle's paper follow:

"Many minerals, metallic and non-metallic, will float on the surface of cold water.

"The behavior is different with hot water. Most of them will float for a few seconds only, becoming wetted and sink, usually a bubble of gas leaving each particle as it sinks. I stop to ponder. What is the significance of this bubble of gas? What does 'wetting' actually imply?

"Crystalline fragments of tourmaline, garnet, blende, galena, pyrite, quartz, gypsum, andalusite and calcite were taken, two of each, one oiled with oleic acid and the other unoiled. These were immersed in water and pressure was reduced. Bubbles immediately formed on the oiled particles; and, on further decreasing the pressure, bubbles larger and in greater number formed on the oiled than on the unoiled minerals. With a lens, minute specks could be seen on the faces of the pyrite crystals, which gradually grew into larger bubbles on reducing the pressure. The bubbles on the oiled particles were of a more permanent character, and the manner of leaving the minerals was different. With the oiled minerals the bubbles became distorted and dragged away more than with the unoiled minerals. On warming the solution the bubbles clung more tenaciously to the sulphides than to the other minerals, the quartz, calcite, and gypsum being almost free from bubbles.

"All minerals absorb gases. Sulphides appear to absorb CO_2 naturally. Hence: (1) Particles are not wholly wetted when immersed in water; (2) particles tend to float when sprinkled on the surface; (3) when immersed in water and heated, the air or gases dissolved in water will collect in particles and float or tend to float them to the surface; (4) saturated solutions of gases in water evolve gas which collects on all particles.

"The flotation of mineral particles appears, then, to depend on the fact that the particles having gas attached to them become entangled, or caught, by the surface film of a liquid. This film may be: (1) the free surface of a liquid; (2) the surface of a bubble of gas under the surface of a liquid; or (3) the surface of a bubble of gas above the surface of a liquid. Once the particles are entangled in this film they are sustained there."

Whence comes the volatile substance that appears as bubbles forming and growing at the surface of mineral particles under conditions of reduced pressure? Do the mineral particles in a froth flotation pulp have a film of gas that has been absorbed from the water or developed by chemical change? Do the chemicals employed in the flotation process furnish or supply the necessary condition for the development of the bubbles, even as reduced pressure apparently functions to allow expansion of the gas film to such a volume that it could be observed and in consequence could impart buoyancy? Do conditioning reagents affect the surface of quartz and other gangue minerals in the same way that hot water did in Mickle's experiments?

One investigator at least, C. T. Durell[10], was not convinced that attachment of mineral particles to mechanically introduced air or gas bubbles is the source of the buoyancy. He contended as follows: "Visible bubbles of air, which have been blown into the water, can in no way sustain the life of a fish. Neither can they aid one iota in dissolving gold in a cyanide solution. Nor can visible bubbles of air introduced into a liquid in this way be attached to solids to aid in flotation. Available air or oxygen for things of this nature must be air actually in solu-

tion. A fish may be breathing freely at the bottom of an aquarium. If the water be warmed so as to expel the air, the fish will rise to the surface and try to jump out. Only nascent hydrogen can unite with arsenic in the Marsh test. In the same way, only nascent oxygen from the dissolved air can act as shown in the chemical equation:

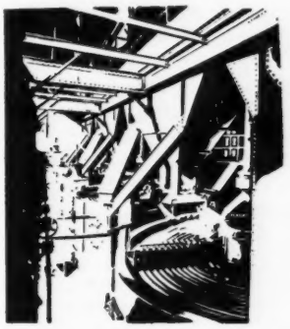
$$2\,Au + 4KCN + H_2O + O = 2\,KAu(CN)_2 + 2KOH$$

when gold is dissolved [in the normal operation of the cyanide process]. Only nascent gas can be attached to mineral in the flotation process."

O. C. Ralston[11] objected to Durell's use of the term "nascent gas." He suggested: "Doubtless Mr. Durell means the dissolved gas that can be liberated from the water, but that is hardly the ordinary sense of the term . . ." Ralston also objected to the word "occlusion," as applied to gases held by minerals. "As I understand it," he said, "there are three ways other than in visible openings by which gases can be held in solids. These are (1) in solid solution, in the same way that gases can be held in liquid solutions; (2) in surface adsorption, where the layer of gas immediately in contact with the solid is found to be held by some force of attraction, unnamed, and is hence considerably compressed; (3) in occluded form; this is a term the meaning of which has been much disputed; it is often used in the sense of one or the other of the terms above mentioned; of late there has been a tendency to call occluded gas any gas that seemingly is held in some manner different from that designated by either of the other two terms and in a manner not exactly understood."

Ralston says further that "Another physical fact that casts doubt on this hypothesis is that occluded gases (and dissolved gas as well) are liberated from the substances occluding them only with difficulty; the last traces of them are still held even when the substance is placed in a high vacuum and heated to a considerable degree. It is as though the molecules of the gas were diffusing out through very small clogged pores. How this tightly held gas could be liberated fast enough to compare with the exceedingly short time which it takes to accomplish flotation of a sulphide particle is difficult to explain physically." A notation at this place in Ralston's article which I made is of interest: "Why not explain it chemically?"

Ralston indicated consideration of the composition of the surface of mineral particles when he stated: "That surface films of adsorbed gas or air exist and may be important I firmly believe, but there is little evidence of any great difference in the amount of this gas on gangue and on flotative minerals." Possibly, then, "surface films of adsorbed gas or air" have an important rôle in the mechanism of flotation. Recent investigations in the flotation of non-sulphides (gangue minerals) point to an optimum condition for the selective flotation of one non-sulphide from others contained in the same pulp by the use of fatty acids as collectors. What, then, is the function of soluble chemicals such as soda ash and sodium silicate? Do they function as agencies that remove that "surface film of adsorbed gas or air" of the non-sulphide that is "depressed" or prevented from floating? Under the optimum condition, do the



fatty acids react with the surface film of the floated non-sulphide, or has that film been removed; and, consequently, if chemical activity is occurring, the reaction must be between the fatty acid and the true surface of the mineral? Still another assumption may be offered: Would the removal of the gaseous film of the floated mineral expose an altered surface that has a selective chemical affinity for fatty acids? Could this chemical reaction result in the formation of a volatile substance under the optimum condition of hydrogen ion concentration? The application of chemical flotation to the treatment of non-sulphides has only begun.

Reference was made by Durell to an abstract in *Engineering and Mining Journal* of Feb. 10, 1906, of a paper by J. Swinbourne and G. Rudorf. Durell's quotation follows: "A way of demonstrating the presence of gaseous envelopes is to sift some powdered substance which easily sinks, such as sand or ferrous sulphides, on the surface of hot water, previously freed from gas by boiling. Bubbles or gas rise from the surface of solid particles. It seems necessary that the gas should be produced at the surface of the particles themselves. The air film always plays an important part; and if the ore is thoroughly washed or boiled in water, to remove the air film, it cannot be concentrated with acid."

T. M. Bains[12] says: "If powdered galena ore, with a limestone gangue, be dropped into pure water, most of the powder will immediately sink to the bottom. As the air inclosed by the particles is expelled gradually, one sees the formation of 'armored' bubbles, some of which may last for days. Here is flotation without oil or acid. If nitric acid be added, the gas bubbles formed by the action of the acid on the gangue will carry up particles of galena." My observation of this phenomenon seemed to indicate that the sulphide is taken to the surface by bubbles formed from its own digestion.

Mickle investigated the gas content of concentrate from Broken Hill. Especially interesting is the fact that samples of gas could be collected by allowing the concentrate to stand, the gas becoming disengaged without the application of vacuum. Following are the reported contents of several of these samples as noted from Mickle's records by Durell: (1)—nitrogen 72, oxygen 2, carbon dioxide 26 per cent; (2)—nitrogen 82, oxygen 2, carbon dioxide 16 per cent; (3)—a sample of concentrate produced by the Horwood process gave nitrogen 95, oxygen 1, and carbon dioxide 4 per cent.

What is the significance of the presence of such a considerable proportion of nitrogen? Surely this does not indicate that air bubbles or even bubbles of carbon dioxide played an important part in the production of this flotation concentrate, for the ratio of the nitrogen and oxygen is out of proportion to the normal content of atmospheric air, and the carbon dioxide lacks quantity as compared to the nitrogen.

Mickle says: "Given a starting point, any gas generated in or introduced into a solution will tend to collect on the surface of particles. If acid is added, the gases will leave the gangue particles and collect on the sulphides; the latter therefore tend to float and the former to remain under the surface of the liquid. When substances in the solution, such as carbonates, are attacked by acids, the gases evolved saturate the liquid and commence to grow on the sulphides. Some sulphides naturally adsorb gases, and such sulphides readily float. The bulk of Broken Hill ores may be taken as examples. Other ores require preliminary treatment so as to alter their surfaces to some extent. This may be done by merely heating to 250 deg. C. or immersing in some solution that will attack the sulphides."

He says further: "The gas obtained in all cases from experimental work, and also from samples obtained from various flotation plants, was carbon dioxide, with varying amounts of nitrogen and oxygen. No doubt most of the gas is generated from gangue particles in the ore; and, being more soluble in the liquid than the other gases, it collects readily on the surface of the sulphides. In all experiments with sulphides, carbon dioxide was obtained on exhaustion. This gas must have been present in the form of a compound easily decomposed by heating or by reducing the pressure, or as a gas condensed on the surfaces of the sulphides and there retained with greater force than other gases present. The carbon dioxide was the last to come off."

Whatever the source of the gas responsible for the buoyancy attained by the mineral particles in Mickle's research, one must conclude, in part at least, from the record of his observa-

tions, that he considered the condition of the surface of the mineral particle of prime importance. The suggestion of Locke, that "it would be a wise thing for us to go back and see what Mickle did," could be expanded with the advice to study the mechanics of present-day flotation in a similar manner, keeping in mind the possibility of the formation of volatile substances by chemical reactions at the altered surface of the mineral particle and the collecting reagents employed.

In my opinion the Elmore vacuum process is of interest in that it effects reduction of pressure as an aid to the flotation of minerals in those instances where a gaseous envelope surrounds the mineral particle. Although by that process a selective flotation of the sulphides from the gangue was obtained, selective flotation of one sulphide from another was not accomplished. In the light of what is now known in selective flotation, one can readily understand why Elmore failed in this respect. Flotation was accomplished in an acid circuit. Consequently, the possibility of the destruction of the surface film of the gangue minerals must not be overlooked.

T. J. Hoover[14] suggested that "It would be interesting to investigate, if by limiting the quantity of gas available for froth production it would be possible to secure differential flotation between lead and zinc sulphides." In discussing the quantity of air available for lifting the mineral particles in the Elmore process, Hoover says: "If the pulp was made up on the ratio of four tons of water to one ton of ore, and if the ore was of the Broken Hill type, this would be four tons of water to about half a ton of sulphides. The water contains 2.2 per cent of volume of air in solution, and it may be assumed that the vacuum releases 2 per cent by volume. Four tons of water is 142 cubic feet; 2 per cent of this is 3 cubic feet of air released for the purpose of floating half a ton of sulphides. Three cubic feet of air loaded with sulphides to an extent just sufficient to remain in suspension would require 3x63 lb. of sulphides, which equals 189 lb. of sulphides, or considerably less than half a ton. There is enough air in a pulp of 1 ton of water to 1 ton of ore to lift 60 lb. of concentrate. As the proportion of water to ore rarely exceeds 6 to 1, and as the ores treated generally yield more than 16 per cent of concentrate, it will be seen that some gas other than that naturally dissolved in the water must be found to effect flotation. This is generally secured by adding limestone to the ore, and then acid at the point where the pulp enters the vacuum chamber."

To know, if data were at hand, how much gas other than the dissolved air was found necessary to develop in order to furnish buoyancy for the remaining portion of the original half ton of sulphides would be of interest. On the other hand, one might find that chemical activity was involved, if the problem were investigated from the standpoint of gas development at the surface of the mineral particle in the presence of reduced pressure, and contamination or other conditions essential for the developed bubble to function as the carrier.

The term "chemical flotation"[13] has been suggested as a better term than oil flotation, for the reason that, aside from the use of soluble oils as frothers, few of the old-time flotation oils are now used. The apparent reason for discarding oils as collectors is evident—they cannot be used with advantage to secure the selective flotation of minerals. Their soluble constituents are not favorable for selective chemical activity.

How did these flotation oils function? Consult the literature and recall what we were taught. The function of the oil was to give the mineral particle a coating of oil, to which the bubbles of gas or air could attach. Agitation, aëration, circulation, and sometimes heat were essential factors. Agitation, aëration, and heat are tools of the chemist in promoting chemical activity. Is it not possible that these same factors were instrumental in bringing about chemical activity between the soluble constituents of the collecting oil in the presence of dissolved air (containing oxygen) and the surfaces of mineral particles?

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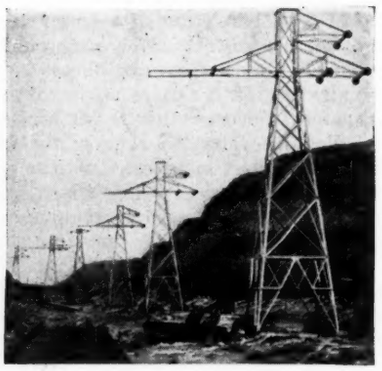
Supporting Trolley Wires in Open-Cut Mine

IN THE experimental installation of electric haulage at its open-cut mine in Bingham Canyon, Utah Copper used portable wooden poles, and also poles made of 4-in. standard iron pipe, cast in a block of concrete, for carrying the trolley wires over the levels. Both means of support proved fairly satisfactory, but in course of time, on account of the necessity for moving the trolley line frequently, the supports of the wooden poles became racked, involving excessive maintenance.

The company later abandoned these methods of support in favor of portable 26-ft. fabricated-steel towers, as shown in the illustration. Towers approximately the same in design as these had been used by the company for carrying the power line to its shovels, previously electrified. The company decided to use similar towers, with modification in design, for carrying the trolley wires also, thus avoiding a second pole line.



Portable steel towers, 26 ft. high, carrying shovel power-transmission line and trolley line for locomotives; Utah Copper mine, Bingham Canyon, Utah

Separate power lines cross each level of the mine, according to J. D. Shilling, assistant to the general manager. Two shovels may be located on one level, in which case the one transmission line serves both. This is a three-phase 5,000-volt line, and on this account cannot be used for operating the electric locomotives. On an arm projecting from the same tower is carried the trolley line, located near the top well away from the transmission line.

In the original design of the tower, the power line running to the shovels was carried directly on top of the tower. In the present form, this design has been modified so as to add the arm which now carries the trolley line. A bracket was also added on the opposite side, near the top of the tower, to carry the shovel power line, as indicated in the cut.

Headframe Design and Construction

Lucien Eaton

THE design and construction of headframes is too large a subject to be discussed in great detail in one article, and I do not intend to treat the subject from a theoretical point of view, but rather to take up the practical solution of the problems that arise when new construction is planned. In nearly every one of the older mining districts a type of headframe well adapted to local conditions has been developed. Its design is usually based on experience, but is often influenced also by what might be termed "heredity," certain features of early practice being retained after the reasons leading to their adoption have ceased to exist.

Materials of construction commonly used are wood, steel and reinforced concrete. In the lighter wooden headframes the members may be built up of plank spiked or bolted together, as is common practice in the Tri-State zinc district, but the usual practice is to use heavy, square timber in sizes from 8x8 in. to 14x14 in. The advantages of timber construction are that it is usually lower in first cost, the material is readily obtained, and the framing and erection can be done by local labor. Its disadvantages are its comparatively short life, even when painted, and the danger of fire. Moreover, the size of headframe that can be constructed of wood is limited.

Steel headframes are taking the place of wood in nearly all mining districts. Small headframes of steel have the advantage of lightness and portability, and, if the design is standardized, may be used over and over again at exploration shafts or mines of short life, the capital expense being thus reduced below that of wooden structures. These small headframes are made of standard rolled sections tied together with angle irons. In large headframes, the larger members may be made of single rolled sections, as the tendency is now, or they may be built up and latticed. The choice depends somewhat on the cost of transportation, the made-up members being lighter and more easily handled. Steel has the advantages of being permanent and fireproof, and it imposes no structural limitations upon the height of the headframe. It also lends itself easily to extensions, should these be necessary. On the other hand, its first cost is usually higher than that of timber, construction and delivery require more time, and special labor must often be supported for erection.

Use of concrete in the construction of headframes has received some attention, but in spite of its advantages as a material for structures of long life, the number of concrete headframes in use is relatively small. Two types of construction have been used: a light steel structure, the members of which are inclosed in concrete; and reinforced concrete, in the form of beams and columns, or as more or less continuous monolithic walls. Concrete possesses the advantages of permanence and rigidity, requires little maintenance, and is fireproof. If the necessary materials can be obtained locally at small expense, it may be cheaper than steel, and it can usually be erected by local labor. It imposes certain limitations in design, however, and, when once built, is not readily changed or extended.

One use of concrete in headframe construction not often seen in the United States is in the construction of a house or room over the shaft, on top of which a small steel headframe is erected. This combination of two types of construction has possibilities of more general development.

One of the simplest designs of headframe, and one very popular for small shafts, is the A-frame, in which one set of vertical members is braced by two back-legs. An example is shown in Fig. 48. This type is easily erected, because the vertical part can be framed and put together on the ground, like a trestlebent, and can be raised as a unit and braced by the back-legs; but, if properly braced against side-sway, the saving in timber over the four-post design is not so great as might at first appear to be the fact.

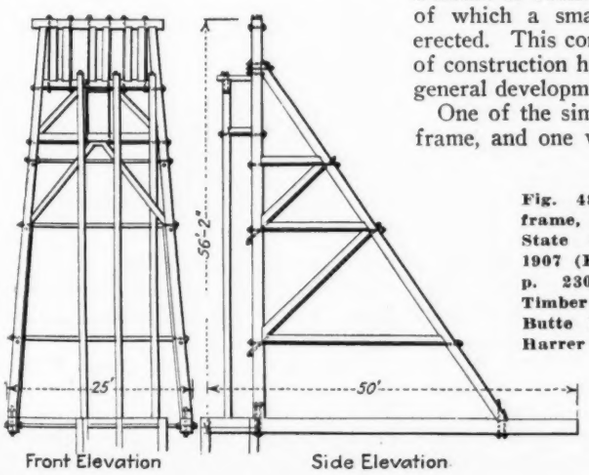


Fig. 48—An A-type headframe, developed at Badger State mine, Butte, Mont., 1907 (E.&M.J., Sept. 8, 1930, p. 230, "Construction of Timber Headframes in the Butte District." By C. S. Harrer and L. S. Farnham)

The four-post design of headframe without back-legs is usually the most economical in respect to material needed. The front legs are spread at the bottom to resist side-sway, and the rear legs are brought back and also spread to take the place of back-braces. With this design, however, the hoist must be placed fairly close to the shaft. The headframe at Roan Antelope's Beatty shaft is a fine example of the four-post type without pockets.

The six-post frame, or four-post frame with back-braces, lends itself so readily to a variety of conditions that it can be used in some form at almost any shaft, whether inclined or vertical. It is particularly suitable for shafts where incorporation of loading pockets in the headframe is desirable. In such an application, the headframe proper really stands upon the bin structure. A six-post headframe is shown in Fig. 49.

A type of headframe from which much was expected some years ago consists of an unstable vertical framework carrying the guides and sheaves, and depending for its stability on guy ropes. This type is subject to much vibration, and is not to be recommended except where transportation is a paramount consideration.

The forces to be resisted by the members of the headframe are principally the stresses set up by the hoisting ropes. The back-braces are customarily placed at such an angle that they are in line with or outside the resultant of these forces, no allowance being made for the weight of the headframe and its resistance to overturning on that account. In heavy structures, especially those of concrete, the weight of the headframe contributes materially to its stability, and may well be taken into consideration in making calculations. Unless the hoist has but one drum, the pull of the hoisting ropes is seldom on or exactly parallel with the center line of the headframe, and in almost every case, when hoisting is in balance, the stresses in the two ropes are never the same and considerable

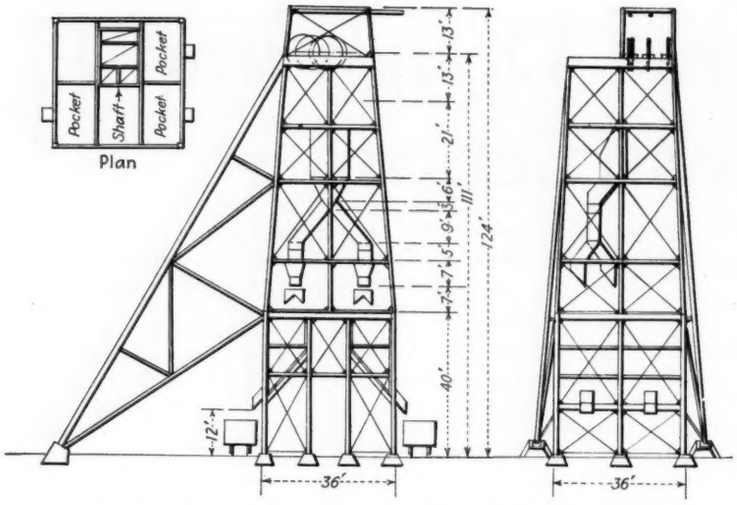
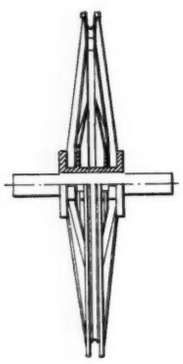


Fig. 49—Four-post headframe with back-braces (or a six-post type) and pockets

torsion is developed, first in one direction and then in the other. The shock of dumping the skips is also important and a source of much vibration.

In open country where winds are high, wind pressure may be an important consideration in design. Under ordinary circumstances it may be neglected, for a headframe strong enough to withstand the stresses and shocks of hoisting and dumping cannot be overturned by any wind pressure to which it will be subjected.

In determining the height of the frame, the designer customarily works from the ground up. If pockets are to be built either in the headframe or adjacent to it, the height at which they must discharge and their volumetric capacity determine their height and in turn the height of the skip-dump. If ore is to be stocked during part of the year, or if height must be provided for disposal of waste, one or the other of these considerations may be the deciding factor in determining the height of the floor at the top of the pockets. If tramming is to be done on this floor, chutes and small pockets must be provided above it for loading the cars, and swinging doors, or "butterflies," or other means of diversion, must be installed to facilitate the separation of ore from waste and of one grade of ore from another. Headroom for all of these must be provided under the skip dump, and every complication in grading ore increases the number of chutes or pockets required. In some forms of construction, where more than four grades of material had to be kept separate, two dumps, one above the other, have been used, the lower being designed so that it could be thrown in or out as desired. In practice, this is an unsatisfactory solution of the problem and should not be used, except as a last resort. In calculating the headroom required, liberal allowances should be made, lack of headroom being a serious handicap in operation.



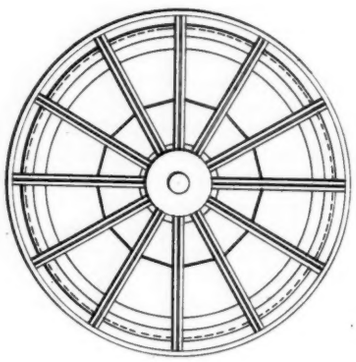


Fig. 50—Electrically welded sheave, patented by Allis-Chalmers

After the height of the discharge point of the skips has been determined, allowance must be made for the travel of the skip in dumping and a further allowance for overwinding between the normal dumping point and the sheaves. This last allowance is sometimes limited by law, but this limitation applies more particularly to man cages than to skips. In good practice it is from 10 to 25 ft.

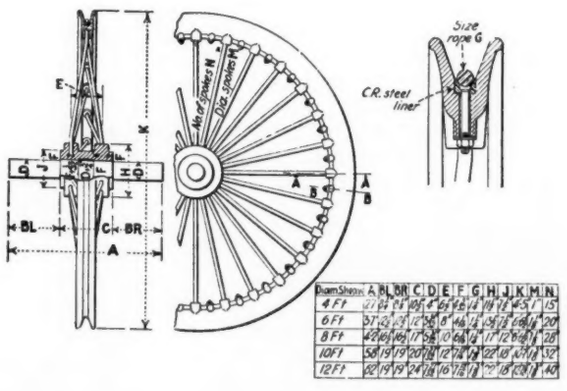
If releasing hooks are used, they will require 3 to 5 ft. of additional headroom. They are used more frequently with cages than with skips, and are not considered essential in the United States. In their stead good practice calls for bringing the guides closer together below the sheave, so that in case of a serious overwind the cage or skip will wedge so tightly between them that it will not fall down the shaft, even though the rope is broken.

To the height of the center of the sheaves must next be added the height of the crane by which the sheaves are set in place. This usually consists of two A-frames supporting an I-beam, on which a trolley runs. The I-beam extends beyond the end of the sheave platform, so that the sheaves can be hoisted up outside the building. This crane must be high enough to raise the sheaves clear of their supporting beams.

The width of the base is to a certain extent a function of the height, and the minimum allowable is determined by the side stresses developed in hoisting. In practice the width of the headframe is often determined by that of the pockets included in the structure.

If we assume, as an example, a vertical shaft equipped with a cage and counterweight and two 5-ton skips, which must handle waste and three grades of ore, the ore being shipped by rail from pockets in the headframe during part of the year and stocked at other times, the dimensions of the headframe would be determined approximately as follows:

Three pockets are required, two on one side of the headframe and one on the other, and the fourth corner may be reserved for waste. The minimum length of two pockets would be 30 ft.,



Diam. Sheave	A	BL	BR	C	D	E	F	G	H	J	K	M	N
4 Ft.	27	13	11	10	4	6	8	4	1	14	12	4	5	1	15
6 Ft.	37	15	13	12	5	8	10	4½	1	15	13	4	6	1	20
8 Ft.	42	16	16	17	5	8	10	6	1	17	12	6	6	1	26
10 Ft.	58	19	19	20	7	8	12	7	1	22	16	7	7	1	32
12 Ft.	62	19	19	24	7	8	16	7	1	22	16	8	8	1	40

Fig. 51—Steel-lined bicycle sheave

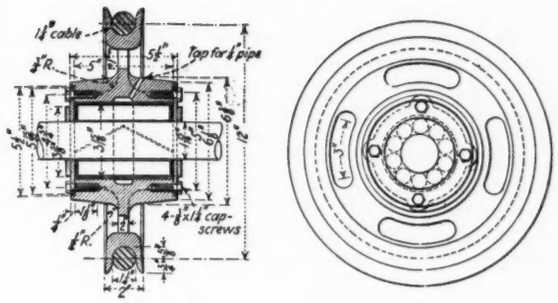


Fig. 52—Cast-steel idler-sheave (E.&M.J. March 19, 1927. "Up-to-date Practice at Cliffs Shaft Mine." Lucien Eaton)

but inasmuch as this is not long enough to include the skip roads and cage road and the distributing chutes inside the building, the dimension should be increased to 36 ft. If we allow 12 ft. as the width of each pocket and 12 ft. for the shaft, the width of the building will be 36 ft. also. The economical height for stockpiles, from which ore is loaded by power shovel, is 40 to 42 ft. This height may therefore be assumed as that of the stocking floor at the top of the pockets. Above this floor headroom must be provided for the stocking cars and their loading pockets, for two "butterflies," the distributing chutes, and the skip dump. The elements making up the height are:

	Feet
Height of stocking floor	40
Clearance for stockpile car and track	7
Loading pocket for stockpile car	7
Butterfly and chute	5
Chute	9
Butterfly	3
Dump chute	6
Clearance under lip of skip	5
Height of skip in dumping position	16
Clearance under sheave axles	13
Total height to sheave axles	111
Crane for sheaves	13
Total height	124

If the width between pockets is increased, the height must be increased also, on account of the slope of the chutes. The headroom required for handling waste is the same as that for ore, if its disposal is at the same elevation. If stocking ore is unnecessary, waste can be hauled upgrade to its dump, and the height of the headframe can be materially decreased.

In cold or rainy climates part or all of the headframe should be inclosed to protect the workmen from the weather. The amount of inclosure needed is dependent upon local conditions and practice, and in cold climates the entire structure from the working floor or landing floor to the top of the skip dump, including the skip roads and cage road, should be inclosed, not only to protect the men but also to prevent freezing in the dumps and chutes. The inclosure is usually made with corrugated iron or asbestos board. Ample window space for lighting should be provided. In concrete structures with continuous walls particular care must be exercised to provide doors or windows large enough to permit the passage of the largest pieces of equipment that must enter the building.

The position and area of the working floors are important and require careful planning. The beams on which the floors are laid must be strong enough to support any load that may be temporarily placed upon them. Three-inch rough plank usually makes the best flooring, and in steel or concrete buildings it should be spiked to 3-in. nailing strips, which in turn are bolted to the beams. Three-inch plank is also the best material for stair treads. All exposed sides of floors and all stairways should be provided with railings. Steep ladders, especially those outside, should have back guards.

In designing the headframe, facilities for putting on and taking off skips and cages should not be overlooked, as sometimes happens. A little careful planning here will save much unnecessary labor later. A jib crane for handling cars and other heavy equipment is a great convenience. Hoisting can be done with a winch or small air or electric hoist.

For convenience in handling and for simplification in design the placing of all the sheaves side by side at the top of the headframe is usually desirable, but where the sheaves for the skips and those for the cage and counterweight would interfere with each other, if placed at the same elevation, the latter may be placed lower down without reducing the clearance unduly.

Sheaves are ordinarily of the bicycle type, having cast-iron hubs and rims. In the best practice the rims have renewable steel liners fitted in the groove, which are machined after being bolted in place. Large sheaves of this type are heavy in the rim, and, if the rope speed is high and deceleration rapid, they may continue to revolve after the rope has come to rest. This causes undue wear on the rope at the stopping points. To overcome this difficulty, the weight is reduced by making the sheave entirely of steel. In one kind of construction, patented by Allis-Chalmers, the members are electrically welded, and, in another, favored for foreign shipment, steel castings are used and the rim is sectionalized, the sections being bolted or riveted together when delivered. Steel rims or liners reduce the wear on the hoisting ropes, and have a life several times as long as cast iron, which usually lasts from three to ten years. Examples of sheaves are shown in Figs. 50 and 51.

The sheave axle is usually pressed into the hub. When conditions require the sheave to be hoisted into place between the bearings, instead of being hoisted outside and then lowered into place, the axle is slipped in by hand afterward, in which case it should be keyed in place. The shaft should be larger in the hub than in the bearings, the difference being 1 in. in the larger sizes, 8 ft. or more in diameter, and half an inch in the smaller sizes, and should be turned with a good fillet. The use of an alloy steel, such as Agathon steel, made by the Republic Steel Corporation, is recommended for sheave axles. It should be heat-treated to a hardness that will just permit machining. Bearings should be designed for a pressure of not more than 400 lb. per square inch of projected bearing area, and should be lubricated with light grease fed by means of force-feed grease cups. Some bearings are made with a grease reservoir in the lower half beyond the end of the shaft, a design that is to be recommended. Ball and roller bearings for sheaves have been tried in a few instances, and the claim is made that through their use the life of the hoisting rope is notably increased and friction reduced.

The diameter of the hoisting sheave should be not less than 72 times the diameter of the rope, if a plowsteel rope is used, and 90 is better, if rope wear be considered. For softer grades of rope a larger ratio should be used, because the elastic limit of the wires is less. In cross-section the groove should be turned to a circle ⅛ in. larger in diameter than the actual diameter of the rope. Length of hub is important, and should be 15 to 20 per cent of the sheave diameter.

To keep the hoisting rope from flapping, it is customarily supported at intervals upon idler sheaves, which are carried by towers or pulley stands. The number of such stands required depends on the distance between the hoist and the shaft, and this in turn depends upon the length of the rope travel upon the drum and the fleet angle allowed. The stands are unevenly spaced, so that the rope will not have rhythmic vibrations, the spacings usually varying from 30 to 50 ft. Under the conditions outlined, the sheaves may be set in a straight line, or, especially in long runs, on a catenary slightly flatter than the natural curve of the rope at maximum tension.

Low pulley stands are sometimes made of timber, framed like trestle bents, and set in concrete. High stands are nearly always made of light structural steel, and may be either self-supporting or braced against the headframe by successive trusses. Platforms are provided for the man who does the oiling. These are reached by ladders, if their height is less than the distance between the stands, and by runways on trusses between the stands when the

height becomes excessive. Ladders, runways, and platforms should be thoroughly guarded.

Shape and size of the idler-sheaves have an important bearing on the rope life. The idlers should be as light in the rim as is compatible with resistance to wear, and of a diameter that will give a turning moment sufficient to keep the rope from slipping and the revolutions per minute within reasonable limits. To reduce both weight and wear, the sheaves should be made of manganese steel or cast alloy steel. Hyatt roller bearings placed in the hub are particularly suitable for the bearings, because the load is light and because the sheave must be free to travel on the shaft. The sheave shown in Fig. 52 was made of cast alloy steel, weighed only 40 lb., and gave satisfactory service with a diameter of 1 ft. at rope speeds up to 1,000 ft. per minute, The life was eleven times that of cast iron, and when worn the groove was easily renewed with an electric arc welder.

Ore should be delivered to the pockets as nearly automatically as possible, and, unless too great headroom is required, distribution should be made by gravity through chutes. Diversion of the product to its proper destination is accomplished in the chutes by butterfly gates, which are usually made of plank, covered on both sides by steel plates. Attached to the lower edge of the gate is a heavy, square axle, turned at the ends, the bearings of which are outside the chute. On the axle is a yoke, attached to compressed-air cylinders, by means of which the gate is operated. The control valves of all the cylinders may be mounted on a central switchboard, if desired. Movable chutes, either hinged or running on a track, may be used to effect the final diversion to the loading pockets, and, if the headroom is insufficient for distribution by chutes, belt or pan conveyors may be used. Transfer by car should be made only when the quantity is small.

Loading pockets or storage pockets may be separate structures, or may be integral with the headframe, and should therefore be considered with headframe design. For each grade of ore a separate pocket should be provided, the capacity of which should be sufficient to prevent delays either in hoisting or in disposal of product under average working conditions. When shipment from the pockets is made by rail, if the daily production is small, two grades may be loaded alternately from pockets on the same track, but if the production is large, a separate track for each grade is very convenient.

Pockets for coarse ore are usually built with sloping bottoms, and have chutes in the side that is parallel with the tracks. The slope of the bottom varies from 40 to 50 deg., depending on the readiness with which the ore runs. Too steep an angle causes blocking at the chute and excessive pressure on the front of the pocket, and too flat an angle prevents free or complete discharge. Ordinarily, the ore is allowed to accumulate on either side of the chute opening, where it acts as a cushion and prevents wear on the front of the pocket. For fine ores that tend to pack, side-loading pockets also are desirable. Blocking at the chutes can be minimized by arching the front outward. The slope of the bottom is steep, varying from 50 to 70 deg., according to the character of the ore. The corners on either side of the chute had best be filled in, care being taken to produce no valleys with slopes too flat.

A side-loading pocket usually consists of a strong framework of steel or wood, inside of which is a lining of plank, partly or wholly protected by steel wearing plates. The front should be stronger than the sides or back, and the plank lining there should be at least 1 in. thicker. The bottom should be stronger still, and here, instead of two layers of plank, 6 x 8-in. timbers are preferable, laid on the broad side, and wearing plates, heavy straps, or rails are bolted to them.

If the ore is crushed before it is put in the pocket, and runs fairly freely, a pocket that discharges at the bottom is often best. Bins of the suspension type are sometimes used, but in ordinary practice the bottom is left flat and the ore is allowed to form its own slope inside, thus preventing all wear of the bottom, except close to the discharge openings. The bottom of the bin must be strongly constructed, and in rectangular bins, where the walls must resist heavy side pressure if the ore column is high, they must be thoroughly braced with tierods. On account of this side pressure, cylindrical bins of steel and concrete have structural advantages. Unfortunately, however, a circular bin is more likely to hang up than a square one. Hanging up is serious in a bin that discharges through the bottom, because poking through the chute becomes almost impossible. If the ore runs freely, however, this type of pocket is almost ideal. Bottom-discharge pockets require few wearing plates, because the ore slides on itself nearly everywhere, and wear is concentrated at the discharge points.

The size of the discharge openings, their position and number, are important. For handling coarse ore the opening should be approximately three times as wide as the grizzly opening through which the largest piece will pass, and should be not less than two-thirds as high as wide. Chute openings, which are merely holes cut in the front wall of the bin, should be 25 per cent larger than those which are at the bottom of a chute or slide. An opening 42 in. square will pass all pieces freely that can be loaded by hand, and an opening 30 in. square is a convenient size for ore that will pass a 10-in. grizzly. Freely running and finely crushed material, such as quartzite or limestone, can be handled satisfactorily through an opening 24 in. square, but openings as small as this should be used only when the size and characteristics of the material to be handled are known to be favorable, because after the pocket is built, the size of an opening can be more easily reduced than enlarged.

The type of gate or chute closer is also important. For coarse material, fingers or undercutting gates, operated by air cylinders, should be used. These work well with fine material also, but are not essential, inasmuch as overcutting arcs or doors work satisfactorily, if the chunks that would prevent their closing are few. For bottom-discharge pockets single or double under-cutting arc gates are satisfactory. They require little power, and are quick and positive. Another type of closer for a vertical chute of this kind consists of a hinged spout, which can be raised until it is too flat to allow the ore to run.

In a pocket not longer than one railroad car and discharging at the side, one opening is usually sufficient. Most pockets have two or three openings, but most of the loading, if not all, is usually done at only one, so that the others represent a needles expense. In a bottom-discharging pocket, two or three openings in a car's length may be advantageous.

At a pocket discharging at the side the near rail of the railroad track is 3 to 4 ft. from the side of the pocket, and the lip of the chute 12 to 14 ft. above the rail and extends to within 12 in. of the line of it, the other side of the track having at least 6 ft. clearance. At a pocket discharging through the bottom the railroads ask clearances of 16 ft. under the chute and 4 ft. outside of each rail.

Although a discussion of crushing machinery belongs more properly under the heading of milling than of mining equipment, the fact that in many instances crushers are installed in or near headframes calls for some mention of the subject.

Jaw crushers have larger openings than gyratories of the same capacity, and weigh less, but they require feeders and produce more vibration. The cost of a feeder plus that of a jaw crusher is usually greater than the cost of a gyratory crusher large enough to take the pieces of the same size. Gyratory crushers produce less vibration, require less attendance and are able to receive the full discharge of the skip without blocking. In sizes with openings up to 12 in. wide, they may be safely erected on the steel of the headframe, the vibration being practically eliminated by reducing the speed 15 per cent below that recommended in the catalogs. Larger sizes should have independent concrete foundations.

Placer Ground Sampled By Well-Digging Equipment

Marshall D. Draper

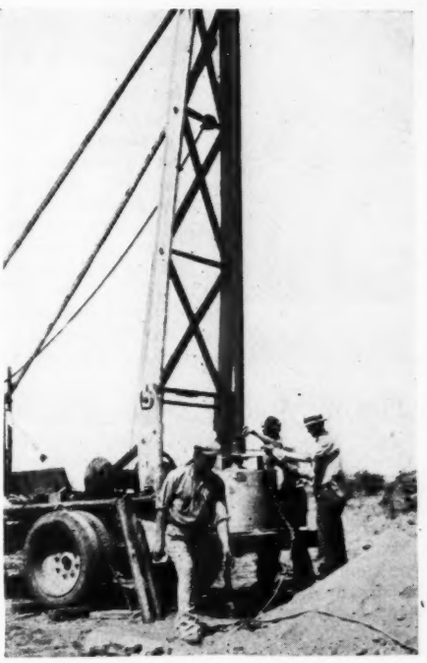
IN A DESERT PLACER in southern California I was recently engaged to examine, the ground, as shown by preliminary tests pits and drilling, stood fairly well without temporary timbering. Average thickness of the gravel was about 25 ft., with extreme ranges of from 12 ft. minimum to 48 ft. maximum. It seldom had any tendency to ravel from the sides, and few rocks were above 6 in. diameter. Conditions were such that some form of mechanical apparatus could be used, particularly because considerable work was contemplated.

An assembled well-drilling machine was obtained for this purpose. It consisted of an automobile truck chassis with sheave ladder bolted to the outside end of the truck and hinged to fold down on top of the cab. When erect, the sheave ladder served to hoist the drilling rods and digging cylinder out of the hole. With rods and cylinder swung clear, the contents of the latter can then be dumped to one side.

The machine proceeds to the site under its own power. The sheave ladder is erected and guyed. Hoisting cable is run over one of the two sheaves and attached to a square "Kelly" or drill rod, and the machine is then ready for operation. Clutches are arranged on the truck body so that, through a ring gear mounted in the end of the truck bed, power is transmitted to the digging cylinder through the medium of the Kelly. The cylinder is usually about 2.5 ft. diameter, or at least of sufficient size to allow work in the hole. The Kelley is centered in the ring gear by a heavy yoke, through which it is free to slide in a vertical direction.

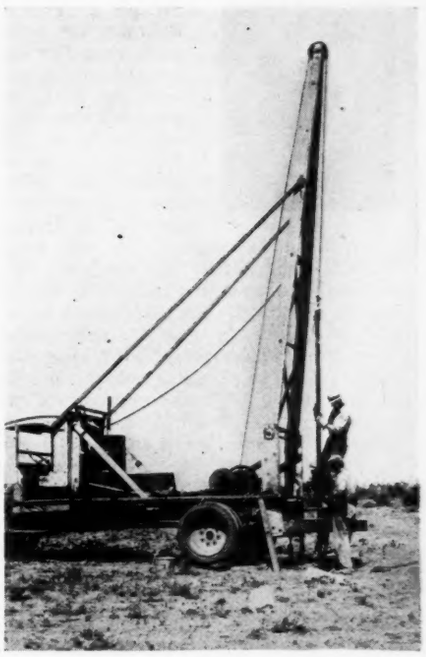
Other clutches are so arranged on the chassis to operate two small independent rope drums. One of these lowers or raises the Kelly and the digging cylinder. The other lowers or raises the necessary additional lengths of Kelly stem, each about 18 ft. long, as the hole is deepened.

The digging cylinder has two opposed cutting blades bolted to its partly open coned bottom, and two more are bolted to and project from the sides of the cylinder about ¾ in., thus giving the



cylinder that much clearance between it and the side of the hole. The whole cone bottom is hinged to the bottom of the side of the cylinder, so that on hoisting the Kellys, the cylinder, and the yoke out of the hole, the cylinder and rods can be swung free and the contents dumped by releasing a latch holding the cone bottom to the cylinder side.

Drilling speed depends largely on the character and number of rocks encountered. If these are not over 3 to 4 in. diameter, a 25-ft. hole can usually be drilled in from two to four hours in ordinary gravel; sometimes in one hour. Otherwise, if large rock is encountered, the operation is delayed by the time it takes to get a man down



the hole, to loosen the rock and send it up. When the hole is deeper than two Kelly lengths, of 18 ft. each, or about 36 ft., the operation becomes progressively slower, because of the need to connect and disconnect the Kellys.

Extremely dry and ravelly material is taken care of by wetting the sides and bottom of the hole, and by bolting sections of old belting to the upper and inner edges of the cutting blades on the cone bottom, thus making a crude flap valve that allows the material free ingress and prevents its egress.

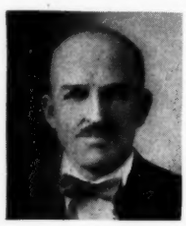
Two men are required to operate the machine. Expenditures other than for labor approximate those involved in operating an automobile of average type, plus blacksmithing costs for sharpening the blades.

The machine has a considerable range of application in drilling dry or semi-dry placers where the depth of gravel is about 50 ft. or less, and where large rocks do not occur in quantity. Uniformity of hole size is an advantage, and the cost per foot of hole is less than if operations were by hand.

Among Contributors To This Issue

Dr. Harrison Schmitt was graduated from the University of Minnesota in 1921, and awarded the degree of Ph.D. in 1926. Subsequent to war service he joined the staff of Standard Oil of Indiana as assistant geologist. In the period 1923-27 he acted as engineer and regional geologist in Chihuahua, Mexico, for A. S. & R. In 1927 he was appointed chief geologist, with headquarters at Hanover, N. M., for New Jersey Zinc, a position held until 1930. In 1931 he established an office as a consulting engineer in El Paso, Tex. Doctor Schmitt has specialized in detailed geological mapping for petroleum and mines; in the direction of exploration work; in the microscopic examination of ores and rocks; and in geological factors in relation to the construction of water-supply services and dams.

M. C. Haas (South Dakota School of Mines) has had an extensive experience in metallurgical work, particularly with gravity concentration and flotation, in Utah (International Smelting), South Dakota (Spokane Silver & Lead), Canada (Treadwell Yukon), Mexico (A. S. & R.). He recently concluded a two-year contract with the Soviet Government, as consultant in the remodeling and operation of mills treating base- and precious-metal ores. In this issue he describes work that led to the adoption of a flotation-cyanidation flowsheet for the treatment of a simple gold ore with which was associated a gangue mineral of refractory physical properties.



F. S. McNicholas (Colorado School of Mines, 1914) gained his first experience, as shift boss underground, with Timber Butte Milling, at Butte, Mont. He has since served on the staffs of Anaconda Copper, in Montana and Idaho, as mining engineer; of Butte & Superior, Montana, as mining and efficiency engineer; of Buena Vista Mining, Cuba, as mine superintendent; of Bar Principal Leasing, Colombia, Central America, as general superintendent. For the past three years Mr. McNicholas has been mine superintendent of the Hidden Creek property of Granby Consolidated Mining, Smelting & Power, British Columbia.

Otis D. Welsch (State College of Washington, 1914) first practiced as flotation operator, and student of M. P. Dalton, at the plants of Federal Mining & Smelting, in Idaho. In 1916 he was associated with Jay A. Carpenter, now professor of mining at the University of Nevada, at the West End Consolidated mill, at Tonopah, where he showed initiative in the development of flotation apparatus. Association with Consolidated Coppermines (Nevada) and Granby Consolidated (B. C.) companies as flotation engineer was followed by cyanide work at Tonopah. In 1920 he was engaged by the Saltchuck Mining company for special research in connection with the flotation of platinum, palladium, gold, and silver in Alaska. He has since practiced as a consulting metallurgist, for the period 1927-31 with Southwestern Engineering Corporation, of Los Angeles.

Marshall D. Draper (Colorado State School of Mines, 1897) has had an unusually wide geographical and technical experience, as manager of mining properties and examining engineer in various parts of the United States, and in Mexico, Alaska, Malay States, China, Burma, and Indo-China. In 1920 he was appointed mining adviser and examining engineer for the Yunnan State Government, and chief engineer for Yunnan Kotchu Tin Trading, positions held until 1925. After acting in an examining capacity for Eagle-Picher Lead in 1926, he returned to China to reorganize the properties of the Yunnan company. He is now vice-president of Yellow Pine Mining, and vice-president and general manager of Pacific Basin Gold Mines.

A Plea for Mature Geologic Opinion

To the Editor:

In the June and July issues of *Engineering and Mining Journal* appeared two admirable articles on gold deposits by C. O. Stee and S. D. Strauss respectively. These two articles, particularly interesting in their major theses, touched briefly and pertinently upon the financing of gold prospects — a subject of major importance in this present depression. Mr. Stee pointed out how the Siscoe property was regarded by many examining engineers as a "wildcat," and, but for the opportune financing of the brokerage house of Stobie-Furlong, would still be a worthless prospect. Mr. Strauss commented as follows: "Bankers want more gold, but they are not willing to furnish the money to get it. . . . If the gold supply is to be maintained, capital must be available for gold-mine prospects."

If the Siscoe incident offered an isolated instance of the mining profession being loath to develop a gold prospect, we might well blame the timidity of capital for lack of new gold mines. This is not true, however. The lack of successful development can in part be attributed to mining engineers and geologists in the field, who act as the representatives of capital. In addition to the Siscoe, the Granada, in the Rouyn area, in Quebec, is another case of a mine resulting from the courage and persistence of a single energetic individual, Mr. Bob Gamble, who brought in this property after a well-known gold mining company had optioned it and carried out a program of supposed barren development. And, recently, the Beattie property, in this same area, was brought to life by Mr. Thayer Lindsley after it had successively been rejected by well-known mining companies. It so happens that Mr. Lindsley was responsible for the initial work done on the Granada, and later came to the help of the Siscoe when it needed more funds for expansion and new equipment. He has also secured an outstanding gold property recently discovered (Island Lake) in northeastern Manitoba, Canada.

That most of the promising gold properties and other mineral discoveries in Canada should gravitate to this Mr. Lindsley seems strange. Is it due to some stroke of hidden geological magic? Perhaps the answer is that, unlike most geologists, he does not specialize in "silicification," "sericitization," "chloritization," or any other "ation." He carries out work on deposits—preliminary work, not an elaborate geological report; then he determines his next step.

Recently I was informed about the conclusions, derived from a brief visit, regarding the Harricanaw section of Quebec by a well-known New York mining engineer. He reported the gold deposits there to be of the shallow-seated type and therefore postulated that they would not persist to any depth. This new Quebec gold district, accordingly, did not merit intensive development. This conclusion recalls those of the early days of Rouyn, Quebec, where equally reputable engineers declared the geology unfavorable for the occurrence of economic deposits — there was too much granite; the quartz was of the glassy vitreous type; the heavy sulphide mineralization was a poor indication of gold; the structure was too tight. And yet, in spite of these handicaps, the Rouyn area provided the bonanza Noranda and lesser economic deposits.

What conclusion is to be derived from the foregoing? Courage, less speculation as to the geological aspects of a deposit in its preliminary stages, and more work before rendering expert opinion, may change many of our ideas—preconceived in most cases—concerning the merit or unworthiness of gold prospects.

ROBERT D. HOFFMAN.

New York, N. Y.

W. H. Trewartha-James is in Alaska.

M. J. John has rejoined the staff of Mount Morgan, Ltd., Queensland, as chief engineer.

Robert A. Boyce has moved his office to Atlas Building, 350 Bay St., Toronto, Ont.

Owen Letcher recently made an extended trip through the gold-producing regions of the Belgian Congo.

George Gray, recently relinquished the position of general superintendent, Mount Isa Mines, Queensland.

David L. Forester recently joined the technical staff of Southwestern Engineering, Los Angeles, Calif., as metallurgist.

Richard Vail has returned to New York after an extensive trip through the mining districts of Ontario and Quebec.

Warren S. Smith has completed two years' work of geological and exploratory examination of the Magnitogorsk iron deposits in Russia, and expects to return to the United States in the near future.

Colorado School of Mines graduates form Engineers Lease, Inc., and operate a property

John B. Huttl recently returned to New York from Continental Europe, and left for Nevada and California.

W. C. Farrier has been appointed Chief Inspector of Mines, New South Wales Mines Department, vice **J. B. Jaquet,** retired.

R. L. Aston has been granted the degree of Doctor of Philosophy by Cambridge University for a thesis on the seismic method of geophysical prospecting.

H. Sommerlatte, of the Institute of Technology, Berlin, Germany, has commenced work as research fellow in ore dressing and flotation at the Montana School of Mines, Butte, Mont.

George H. Garrey, who met with an automobile accident near Pioche, Nev., on June 4, resulting in fractured ribs and injury to the spinal column, and who has been confined to his bed in a Denver hospital, is convalescing satisfactorily.

Charles Grabowski, general superintendent for Oliver Iron Mining in the Eveleth district of the Mesabi range for many years, was retired on pension recently. **W. H. Plummer,** of Marble, Minn., will take Mr. Grabowski's place at Coleraine.

C. W. Van Law has returned to New York City from a visit to the interests of the A. S. & R. Company and Mining Trust, Ltd., to the Mandated Territory of New Guinea, where Mining Trust has important holdings and where the Bulolo Gold Dredging company is operating. Access to the goldfields was made by airplane from the east coast—the only method of transport—and return, to the west coast, was made by flight across the island of New Guinea. Prior to this investigation Mr. Van Law, for 4½ years, was in charge of engineering work in South America for A. S. & R., and manager of Northern Peru Mining & Smelting, its subsidiary in Peru.

I. W. Morley was recently appointed by the Mines Department of the Administration of the Territory of New Guinea as Inspector of Mines and Machinery and Assistant Warden, Morobe Goldfields.

Guy C. Riddell, who recently visited the Tchiatouri manganese mines in Georgia, Trans-Caucasia and the Mikitovka quicksilver plant in Doubass Basin, expects to return to the United States this month.

Gerald G. Dobbs, general superintendent of the ore mines and quarries of the Tata Iron & Steel Company, India, recently addressed members of the Canadian Institute of Mining & Metallurgy at Montreal.

C. A. Banks is back in Vancouver after an absence of seven months, during which time he visited New Guinea, Australia, and England. While in London Mr. Banks contributed an important paper on "Air Transportation of Gold Dredges," which was read at a meeting of the Institution of Mining & Metallurgy.

Board of Strategy of Engineers Lease, Inc., discussing mine development plans

Bertram Wood, for many years chief mechanical and electrical engineer at Rhodesia Broken Hill, is now in India on the staff of Burma Corporation.

Howard R. Ward, who has been superintending the operation of chrome mines in New Caledonia for the past four years, is expected back in the United States in October.

Harold K. Shirrefs and **J. Cornelius,** of the Rosebery staff of Electrolytic Zinc, are at Mount Morgan, Queensland, assisting in mill experimental work being carried out under a grant from the Commonwealth Government.

N. K. Karchmer writes to correct an omission from his article on "Floating Apatite in Russia," in the August issue. Mention should have been made of the fact that the Denver Equipment Company made laboratory tests that were of considerable assistance and encouragement to the Russian engineers commissioned to design the plant. In the laboratories of the General Engineering Company, Salt Lake City, tests were made under the supervision of Mr. Ernest Gayford, which definitely predicted the successful flotation of the ore. In collaboration with the Genter Thickener Company, General Engineering officials also contributed much to the design of the plant, lending valuable assistance in the selection and purchase of equipment.

Roy Drier, an instructor in the metallurgical department of the Michigan College of Mining and Technology, engaged in research work on iron ore benefication at the University of Michigan, has been elected to Sigma Xi at the University of Chicago. The honor was the outcome of research work on gold and other rare metals at the Ryerson Laboratory of the University of Chicago early this year.

Otis D. Welsch concludes a letter recently received with comments that form a postscript to his article in this issue. "I think of selective flotation," he says, "of one mineral from another in terms of selective chemical reaction. For example, in some research I recently did on the flotation of sillimanite and dumortierite I found that there is an optimum condition of alkalinity with either soda ash or sodium silicate, or a combination of both, at which these minerals will float selectively from the quartz and rutile gangue. If too high an alkalinity prevails, the sillimanite or the dumortierite float reluctantly, sometimes hardly at all. In other words, the hydrogen-ion concentration must be correct to obtain the selective chemical reaction necessary for the development of that factor required so that the mineral will float. Hydrogen-ion determinations are the governors for the flotation of minerals. The importance of this cannot be stressed too strongly."

Stanly A. Easton, vice-president and general manager of Bunker Hill & Sullivan Mining & Concentrating, operating at Kellogg, Idaho, has been appointed a member of the committee, headed by Alfred P. Sloan, president of General Motors, to consider the status of intergovernmental debts.

The Secretary of the U. S. Civil Service Board of Examiners, Washington, D. C., announces that information is available in regard to the appointment of one assistant assayer in New Orleans, La., and another at Helena, Mont., applications for which must be filed not later than Oct. 27, 1932.

Obituaries

David Lemmon, Utah mining engineer and operator, at Salt Lake City, on Sept. 3; aged 63.

John Watson (Newcastle College of Science, Durham University), a chemist and assayer widely known and respected on the Witwatersrand, an occasional contributor to *Engineering and Mining Journal;* at Johannesburg.

Frederick H. Hatch (University College, London, and University of Bonn), past president of the Institution of Mining & Metallurgy; at London on Sept. 21, aged 68. Doctor Hatch, after service with the Geological Survey of Great Britain and South African Trust & Finance, assisted John Hays Hammond in his examination of Rhodesian mines in 1894. An exceptionally broad professional experience was subsequently gained in Spain, Transvaal, Canada, India, Abyssinia, Siberia, and Natal. In the last year of the War and immediately after the Armistice he reported on the iron-ore fields of Northwestern France, and the iron and steel plants of France and Belgium in the occupied parts of Germany. Doctor Hatch was also an author of note on mineralogical and geological subjects.

Albert F. Schneider (Columbia School of Mines and Freiberg), for many years consulting engineer to American Smelting & Refining, responsible for results of expert metallurgical work in connection with design and operation of smelters in the United States and Mexico; at Montclair, N. J., Sept. 7, of pneumonia; aged 78. Mr. Schneider was born in Cincinnati, the son of a physician and one of the "heroic women" whose services during the Civil War are on record in the Red Cross building in Washington, D. C.

James A. Shepherd; assistant general manager, Roan Antelope Copper Mines; at Luanshya, Northern Rhodesia; aged 43. Mr. Shepherd, a graduate of the Rose Polytechnic Institute, was at one time general superintendent of the Tucson Gas, Electric Light & Power Company, Arizona. From 1927 to 1929 he was secretary to the manager of the Copper Queen Branch of the Phelps Dodge Corporation, being transferred to Northern Rhodesia in the latter year.

Edwin Young, safety and welfare engineer for Anaconda Copper since 1920; at Great Falls, Mont., Aug. 21; aged 79. Mr. Young emigrated from England in the early eighties, and had served in many capacities as a labor leader and arbitrator. His association with Anaconda Copper resulted in significant achievements in the field of accident reduction and employee welfare. His high principles and pleasing personality endeared him to all with whom he came in contact.

Thomas Wright, at Bisbee, Ariz., on July 30, aged 68. Mr. Wright, who had been a resident of the Bisbee district for 42 years, was foreman for Copper Queen Mining when the Spray shaft was sunk.

Dr. George I. Adams, professor of geology and mineralogy, University of Alabama; at Tuscaloosa, Ala., Sept. 8; aged 61.

Edwin E. Chase, mining geologist, graduate of M.I.T., a native of Lowell, Mass.; at Glendale, Calif., Aug. 12; aged 74. Mr. Chase, who established in Colorado in 1880, was an active figure during the boom days at Central City, and was prominently identified with the development of the Camp Bird mine, at Ouray, Colo.

Harry F. Fay, director of Calumet & Hecla and Hecla Consolidated Copper, president of Mayflower-Old Colony; at Poland Springs on Aug. 10. Long connected with mining ventures in the Michigan copper district, Mr. Fay was a pioneer in exploratory and development work in the district. He was president of Centennial Copper and Allouez Mining before those properties were taken over by Calumet & Hecla.

Ulysses Sherman Grant, professor of geology at Northwestern University since 1899; at Chicago, after an operation; aged 65. Doctor Grant, son of Gen. L. A. Grant, of the Union Army, was educated at the University of Minnesota (B.S., 1888) and Johns Hopkins (Ph.D., 1893). He was assistant state geologist of Minnesota from 1893 to 1899. He had contributed much material to the publications of the state and federal geological surveys, and was associate editor of *The American Geologist* from 1897 to 1906.

Robert E. Cranston (Michigan College of Mines), a pioneer in the gold-dredging industry of California; at San Francisco, Calif., July 30. Mr. Cranston was best known because of his work in the design and operation of gold dredges, and the sampling and valuation of alluvial deposits, in Brazil, Colombia, New Zealand, Australia, as well as in the United States. At the time of his death he was director and general manager of South American Gold & Platinum, and Rimu Gold Dredging, the latter a New Zealand enterprise.

The shaft sinking crew of West Rand Consolidated, Johannesburg, Transvaal, who in May, 1932, succeeded in bettering their own record achieved in March of 400 ft. Both records were made with C.P. No. 5 sinker drills. The officials in the center of the front row are (right) C. S. McLean, general manager; and (left) T. R. Boyde, master sinker.

Five Volumes for the Student And the Practicing Engineer

A.I.M.E. Series. Vol. I: *Choice of Methods in Mining and Metallurgy.* Vol. II: *Mineral Economics: Brookings Lectures.* Vol. III: *History of American Mining.* Vol. IV: *Examination of Prospects.* Vol. V: *Technical Writing. New York: McGraw-Hill Book Company; price $10 the set if purchased before Dec. 31, 1932.*

These volumes were prepared primarily for the benefit of junior members of the A.I.M.E. under the provisions of the Seeley W. Mudd bequest. The first, on choice of methods, records the experiences in particular branches of mining, metallurgy, and promotion by Pope Yeatman, R. E. Tally, F. W. Bradley, E. deGolyer, H. N. Eavenson, L. D. Ricketts, A. L. Walker, E. McAuliffe, N. G. Alford, and A. B. Parsons. Vol. II, edited by F. G. Tryon and E. C. Eckel, comprises lectures, delivered under the auspices of the Brookings Institution, by H. F. Bain, E. C. Eckel, J. R. Finlay, J. W. Frey, J. W. Furness, D. F. Hewett, C. E. Julihn, C. K. Leith, L. Logan, G. F. Loughlin, R. S. McBride, T. T. Read, E. B. Swanson, F. G. Tryon and F. E. Berquist. It is an outgrowth of efforts of the Brookings Institution to stimulate discussion of the problems of mineral economics. Vol. III, by Dr. T. A. Rickard, now a consulting engineer to the United States Bureau of Mines, is a historical account of mining in the United States, providing an interesting record and a valuable work of reference. Vol. IV is a second edition of Gunther's "Examination of Prospects," revised by R. C. Fleming. Vol. V is a reprint of the third edition of Doctor Rickard's well-known treatise "Technical Writing," first published by John Wiley in 1920, to which the author has made minor corrections.

Textbook of Metallurgical Problems. *By Allison Butts. First edition. Pp. 425; illus. New York, N. Y.: McGraw-Hill Book Company; $4. London, England: McGraw-Hill Book Company, Ltd.*

This is essentially a textbook for the college student. Although not intended to replace Joseph W. Richards' "Metallurgical Calculations," written principally for the practicing metallurgist, it contains much new and revised data not included therein. Those accustomed to the use of Professor Richard's book should therefore find the new work useful and valuable.

Application of thermodynamics to chemical and metallurgical reactions, a subject given little attention ten to fifteen years ago, is now arousing sufficient interest to warrant treatment. Among other subjects covered are combustion, heat balance, and production of blast and draft; calculation of smelter charges; drying, calcining, roasting, and sintering; copper smelting; lead smelting; hydrometallurgy, and electrolytic processes; volatilization of metals in melting; and distillation processes, transmission of heat. A comprehensive chapter has been devoted to the metallurgy of iron and steel.

Of the problems for solution used throughout the book to illustrate the various subjects, many represent actual operating conditions at stated plants. Although not intended as a source of definite information as to routine accepted practice at these plants, such treatment should stimulate the student's interest.

Of the numerous tables presented in the final chapter many present data in a convenient form not available elsewhere; most of these data have been taken from reliable reference works, mainly from "The International Critical Tables," and Landolt-Börstein's "Physikalisch-chemische Tabellen."

B. H. Strom.

✦

Theory of The Tectonical Gravimetric Investigation. *By J. A. Mekel. Translated from the Dutch by M. Walter Pesman. Prepared through the Department of Mechanics and Geophysics, Colorado School of Mines, Golden, Colo.: $2.*

This article, translated in mimeographed form, primarily for the benefit of the class in potential functions, presupposes a knowledge of torsion-balance theory. In the first chapter, however, a short résumé is given of the theory, paying especial attention to the significance of observed figures. The author then discusses equations for the potential of gravity, the position of planes of maximum and minimum curvature, and gradients. In the second chapter he discusses mathematically the influences that bodies unlimited in one direction may have on the readings, and derives formulas for the appropriate expressions. In the next chapter he proceeds to the more complicated occurrences of limited bodies, and calculates the formulas for their influences. An example of the application of these principles is given in the final chapter, wherein he discusses a torsion-balance survey he made over the Trias horst, near Winterswijk, to prove the possibility of the existence of such a horst and to determine its south wall. As a result, he develops a profile analogous to that assumed by Van Waterschoot van der Gracht for this horst.

This paper will appeal only to those with a thorough grounding in mathematics, especially potential functions. Others should not attempt to use it. The discussion is mathematical, the paper consisting mostly of formulas.

Sherwin F. Kelly.

✦

Prospecting for Gold. *By Ion L. Idriess. Third edition. Sydney, Australia: Angus & Robertson, Ltd.; 5/-.*

A review of the first edition of this book appeared in December, 1931. The sale of 6,000 copies indicates the meeting of a demand for information on panning and cradling; sluicing and sluice boxes; dams; water power; alluvial or placer gold; dry-blowing; and prospecting old river beds. This edition has new chapters on the prospecting and development of veins; erecting a small stamp mill and operating it; also cleaning up old mill sites; cyanidation; opal mining; and the use of hydraulic elevators.

The Australian opal, one reads, occurs in desert sandstone. Two varieties are known—light and black. If a "stone" is suspected, strike it, and if it cracks or breaks like coarse glass it is probably "potch" or worthless opal. Shaft sinking to opal-bearing ground, and gouging it when "on color," are described and sketched.

This book, with glossary and index, is replete with good methods, described simply. Lack of space forbids quoting the terse directions, and, one may say, conversations.

M. W. von Bernewitz.

✦

Sampling and Estimation of Ore Deposits. *By Charles F. Jackson and John B. Knaebel. Bulletin 356, U. S. Bureau of Mines, Washington, D. C. Pp. 155; illustrated.*

This is a summary of existing practice at United States mines in the main, with occasional references to properties in Canada and Mexico. Essentially, the book is a cross-section, with respect to sampling and ore estimation, of the Bureau's recent admirable series of papers on mining methods and operating practice, although many references are also made to papers outside of this series. Each division of the subject is prefaced by a general discussion of prevailing practice, with suggestions at times as to most desirable procedure, which is then supported by copious citations from the Bureau's papers and other technical literature. The book will be useful to anyone having sampling and estimation to do. Those who already have the mining-methods papers will find it a useful supplement.

TEXTBOOK OF MINERALOGY. *E. S. Dana. Fourth Edition, revised and enlarged by William E. Ford. Pp. 851; illus. New York: John Wiley & Sons; $5.50. London: Chapman & Hall.*

This book follows the third edition by approximately ten years. Although several new texts on mineralogy have appeared in this period, none of them suffice to supplant Dana, a new revision of which is now warranted by the disclosures of a decade of investigation. Among the changes is the addition of a twenty-page section on the mode of occurrence and origin of minerals. The section on descriptive mineralogy, which occupies almost half the book, has been made as complete as possible with respect to all known mineral species, to the extent of giving brief mention at least to each, including even the doubtful or discredited. As a reference work, the book remains unsurpassed.

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PROSPECTING AND EXPLORATION. *By R. C. Matson. Department of Mining, Michigan College of Mining and Technology, Houghton, Mich. Price 60c.*

The contents of this 45-page, 18,000-word, paper-bound pamphlet have been prepared by the author, who is associate professor of mining and civil engineering at Houghton, for the purpose of presenting the principles and practices of prospecting and exploration in a way adaptable to a freshman course in mining. The treatment of the subject is logically arranged, the language of the discussion is simple and clear, and the illustrations are adequate and well chosen. On the whole, the pamphlet represents a job very well done. A good many mining graduates, as well as the freshmen for whom it was written, would be helped by its simple exposition of the elements and apparatus of geophysical prospecting, which takes up the last twenty pages.

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MIAMI-PICHER ZINC-LEAD DISTRICT. *By Samuel Weidman, with a chapter on mining methods by C. F. Williams and a chapter on milling by Carl O. Anderson, Norman, Okla. University of Oklahoma Press. $2.50.*

A need has long existed for a concise but comprehensive technical report on the Miami-Picher zinc-lead district. This treatise fulfills that need. The author, professor of geology at the University of Oklahoma, whose major contributions in the text are the chapters devoted to the geology and ore deposits of the field, has drawn on his intimate study of the district that extends over more than ten years. C. F. Williams and Carl O. Anderson, respectively, write authoritatively on mining methods and milling. Additional chapters consider the history and production of the district, as well as the outstanding features of the various mines. The report closes with a bibliography that lists the more important published references to the district, and classifies them under the headings of geology, mining, milling, statistics of production, and underground health conditions.

✦

SOUTH AFRICAN MINING AND ENGINEERING YEAR BOOK. *Pp. 356; illus. Johannesburg, South Africa: S. A. Mining Journal Syndicate, Ltd.; £1-1. London, England: Argus South African Newspapers, Ltd.; 22s. 6d. (post free).*

A complete summary of South African mining, including production, capitalization, and equipment data for individual companies, as well as general historical information and authoritative discussions of the technical phases of the industry, is presented in this volume. The value of the book to the mining engineer is enhanced by three large maps. One of these shows the various mining properties and transportation lines in the Witwatersrand district; another the coal and iron mines of the Union of South Africa; and the third the power-supply stations of South Africa.

✦

SOUND PRINCIPLES OF MINING. *A report by the Mining Committee of the Los Angeles Chamber of Commerce. Obtainable on request from the secretary, Los Angeles Chamber of Commerce, Los Angeles, Calif.*

A pamphlet of inestimable value to the would-be investor in mining enterprise. *Inter alia,* it emphasizes the following facts: (1) High-grade ore does not indicate a good mine or that the operation of a mine will be profitable; (2) the best paying mines of the world have been those of low grade; (3) a property that has produced and was then abandoned may not be susceptable to reopening and profitable operation; (4) a mine may adjoin a big producer, but this is no justification for the assumption that it can be made to pay profits; (5) large acreage is not necessary in a mining property—many of the great mines have been found in small areas; (6) because one mine has paid a large dividend, this offers no reason why another mine will do the same; (7) the value of a mine can be determined only after examination of the property by a competent mining engineer, who bases his report on facts gathered on the ground, which can be verified; (8) beware of the careless use in a prospectus of scientific terms such as "colloidal," "subatomic," "allotropic," "amorphous," and "volatilization"; (9) it does not follow that because a man has been successful in a commercial business he will be successful in the operation of a mine; mining is a highly technical enterprise that should be undertaken only by thoroughly trained operators and engineers.

RHODESIAN MINING AND ENGINEERING YEAR BOOK. *Pp. 132; illus. Johannesburg, South Africa: S. A. Mining Journal Syndicate, Ltd.; 10s. 6d. London, England: Argus South African Newspapers, Ltd.; 12s. 6d. (post free).*

This volume contains a comprehensive review of the various mining operations in Northern and Southern Rhodesia, as well as a description of the new Benguella Railway and the properties of the Union Minière, in the Belgian Congo. The book is well illustrated, and replete with material of reference value to the mining engineer.

STANDARDS YEARBOOK (1932.) *Compiled by U. S. Bureau of Standards, Washington, D. C.: Superintendent of Documents; $1.*

Subjects covered in this book range frome accident prevention to zoning systems, and include references to work bearing on reducing cost to the public and increasing profit of manufacturers. Other sections contain reports of the work of standardizing bodies within the Federal Government and of states, counties, and municipal agencies, as well as technical societies and trade associations.

NATIONAL DIRECTORY OF COMMODITY SPECIFICATIONS. *No. 130. U. S. Bureau of Standards, Washington, D. C.: Superintendent of Documents; $1.75.*

In this book are listed standards and methods of test for all commodities regularly produced in the United States. Each specification is also summarized as to technical characteristics, scope, and special applications. The ten major divisions of the directory include one on non-metallic minerals and another on metals and metal products.

ZINC AND ITS ALLOYS. *Bureau of Standards, Washington, D. C. Circular No. 395. 214 pp. Price 70c.*

This circular deals primarily with the physical properties of zinc and its alloys. A few production statistics, brief descriptions of manufacturing methods, and extensive bibliographies have been included.

WARNING AGENTS FOR FUEL GASES. *By A. C. Fieldner, R. R. Sayers, W. P. Yant, S. H. Katz, J. B. Shohan, R. D. Leitch. Monograph 4, U. S. Bureau of Mines, Washington, D. C.*

GEOLOGY AND ORE DEPOSITS OF ROUYN-HARRICANAW REGION, QUEBEC. *Memoir 166. Pp. 314; illus. Ottawa: Department of Mines; 45c.*

AEROFLOAT FLOTATION REAGENTS, THEIR PROPERTIES AND APPLICATION. *Technical Paper No. 19, American Cyanamid Company, New York, N. Y.*

ANNALES DES MINES DE BELGIQUE, 1931. *Vol. 32, Part 2. Belgium: Administration des Mines, Brussels.*

INDUSTRIAL PROGRESS

Power-Shovel Equipment Has Reached High Degree of Perfection

J. E. Borland
Engineering Department,
Westinghouse Electric & Manufacturing Company,
East Pittsburgh, Pa.

THE first application of electric power to large excavating shovels was made about twenty years ago. Progress was slow. Operators hesitated to adopt what was considered a new and an untried system, but increase in use of electric power for the operation of shovels has been marked in recent years, from the 15-cu.yd. stripping shovel down to ¾-yd. or smaller.

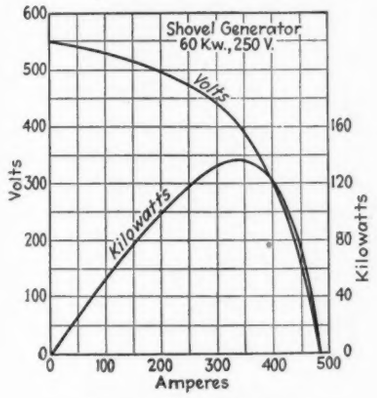


Fig. 1 — Volt-ampere characteristics of variable voltage shovel generator; 60 kw., 250 volts

An individual motor for each of the three main motions of the shovel avoids friction clutches or similar mechanisms that are difficult to maintain under severe operating conditions. In the larger machines, two or more motors may be used on any one motion, but these are operated as a unit from one control.

The electrical system now employed on all but the smaller sizes of shovels uses direct-current motors supplied from a motor-generator set mounted in the shovel cab, using a modification of the variable-voltage scheme of control. Three generators are employed, one for each of the main shovel motions, and control is effected entirely by variation of the fields of the direct-current motors and generators. As the field currents are of comparatively small value, small contacts only are necessary for their control, and the three drum controllers manipulated by the operator are accordingly small and handled with little effort. This is important on equipment requiring continual manipulation of the controls. Stiff and unwieldy apparatus soon tires the operator. To this end, a few auxiliary contactors are sometimes employed to handle the heavier field current of large shovels.

Each of the shovel generators is designed with a separate excited shunt field, a self-excited shunt field, and a differential-series field, the total effect of the three being to produce a volt-ampere characteristic similar to that shown in Fig. 1. This curve shows that the generator voltage is brought to zero at a predetermined maximum armature current (480 amp. in this case), which fixes the maximum torque that can be developed by the motor receiving the power. This limits the strains thrown on the mechanical equipment, and economizes in general maintenance of the shovel.

Generator design is proportioned so as to produce a peak output of about twice the normal rating, with a high output over a comparatively wide range of currents under which the generator will operate in normal service. This insures rapid acceleration and general "snappy" operation.

Fig. 2 shows the arrangement of motor-generator set used on shovels of about 2-cu.yd. capacity, the swing and thrust generators being combined at one end of the set and the hoist generator with the induction driving motor at the other end, the halves joined by a flexible coupling on a common bedplate. Fig. 3 shows the arrangement of an M-G set for a shovel of about 8-yd. capacity. This employs a synchronous motor, and has the exciter direct-connected at one end of the set. Because of the larger size, the set is carried on two separate bedplates and the shafts are joined by a flexible coupling. Motors employed are of the type known as "mill motors." An open-type motor such as that shown by Fig. 4 is frequently used for the hoist and swing motion. The thrust motor, mounted on the boom, is almost entirely inclosed, for protection against the weather.

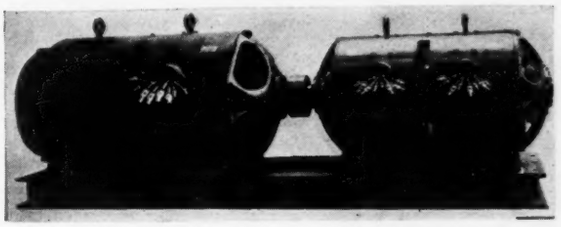


Fig. 2 — Variable voltage motor-generator set for small power shovel

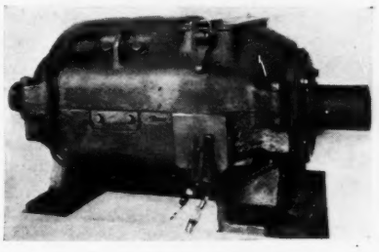


Fig. 4 — Protected-type mill motor

Drum controllers for the three motions are made as small and compact as possible, to place the handle within easy reach of the operator and to offer the least obstruction to vision. A foot-operated controller is occasionally employed for the swing motion, leaving only two handles to be manipulated, these moving in a straight line, forward and back. The appearance of these controllers is shown in Fig. 5. On the smaller shovels, where space is at a premium, starting equipment for the a.c.

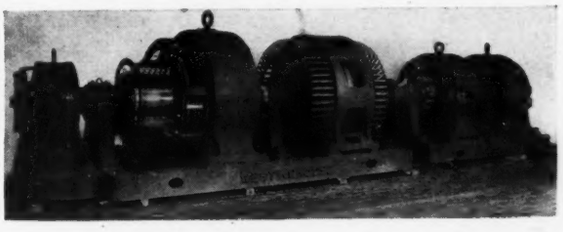


Fig. 3 — Variable voltage motor-generator set for large power shovel

motor driving the main set is ordinarily a three-pole oil circuit-breaker of the self-contained form, which may be mounted on the frame in the most convenient location. On larger equipment a steel switching cubicle, such as that shown by Fig. 6, is generally applied. All circuit breakers, auxiliary power transformers, or other apparatus containing oil are made with special tight-fitting covers and bushings to avoid leakage of oil.

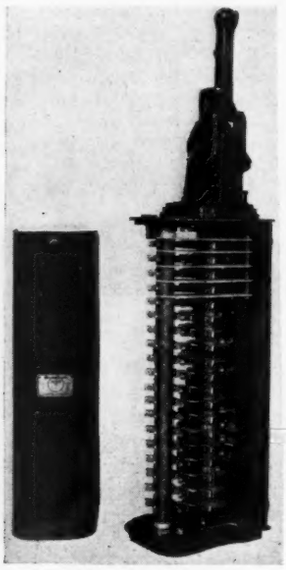


Fig. 5—Controller of variable voltage shovel

Fig. 6 — Alternating-current switching cubicle (sides removed

A variety of auxiliary electrical equipment is employed, including transformer for low-voltage lighting, motor-operated dipper trip, magnet valves for control of compressed air to brakes, and magnet-operated brakes on the motor shafts. All of these are applied as required on the different models.

In shovels below 2-cu.yd. capacity, where the cost of power is not an important item in operation, the advantages of the variable-voltage system are not so marked, and a single direct-current generator is sometimes employed to supply the three shovel motors. The motors, under the arrangement noted, are controlled to a certain extent by variation of resistance in series with the armature, but field control is also employed, which retains some of the advantages of the variable-voltage system in limiting the maximum torques that can be developed, and reducing the amount of series resistors needed. Units of this sort frequently employ a gas engine or oil engine to drive the d.c. generator, thus obtaining the flexibility, low maintenance, and ease of operation of electric drive, with economy of operation and a self-contained power supply. Shovels of this size are also frequently driven by alternating-current motors with rheostatic control. As the motors are comparatively small in size, the drum controllers required, with either direct-current or alternating-current motors, are small enough to be easily operated.

The units described in the foregoing are also applied to draglines, dredges, or other forms of excavating machines. The systems represent the latest practice, the result of careful development, and have been brought to a high state of perfection. That in flexibility, economy, and ease of control these forms of drive are far beyond any other now available is generally conceded.

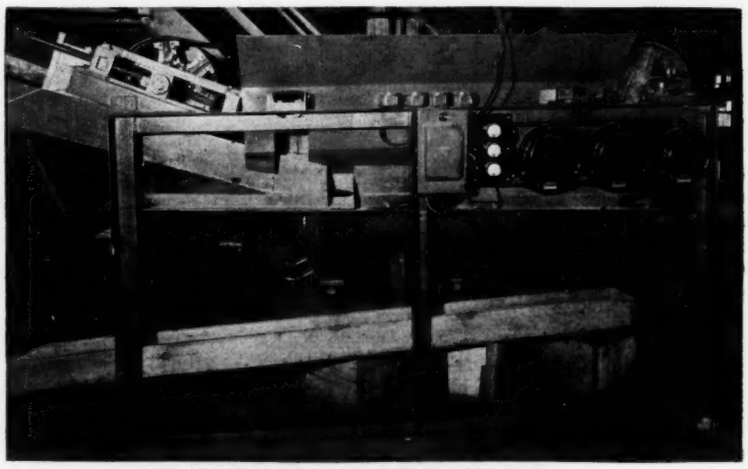
■ ■ ■

Efficient Magnetite Iron-Ore Concentration With New, Wet Magnetic Separator

A new type of wet magnetic separator for the concentration of magnetic iron ore has recently been installed in one of the New Jersey concentrating plants. This is the Crockett-type separator and is manufactured by the Dings Magnetic Separator Company, of Milwaukee, Wis. It is shown in the accompanying illustration. The outstanding feature of design lies in the ability of one machine to make a high-grade concentrate, a clean reject, and a true middling. The machine will handle material from four mesh to the finest dust with practically no losses in the reject, although sizing is desirable both from a capacity and a performance basis.

In making a true middling and a concentrate of the larger sizes of ore, the crushing equipment required is reduced and the concentrate contains that proportion of larger sizes so desirable to the sintering plant. When working on minus 200-mesh feed, the capacity is high and the losses of magnetic ore are practically nil.

The design is such as to throw the phosphorus into the tailing as soon as it is free from the ore. No rapidly moving parts nor high-pressure water jets are employed. In this separator the ore follows its proper path and the machine presents a pleasing appearance at all times. The concentrate is discharged in a sufficiently dewatered condition so that thickeners are not required and can be sent directly to cars. The amount of water required is the minimum needed to carry the middlings and tailing away from the machine.



A finished concentrate, a true middling and a clean tailing are produced in one unit by this new wet belt-type magnetic separator, working on magnetite iron ore

New Explosive More Effective for Breaking Rock

Apex No. 1, a new high explosive of the low-density type for breaking rock, has been announced by the Atlas Powder Company as an economical substitute for the more expensive, high-

strength explosives heretofore used in quarrying. It has been developed to provide an explosive of the dynamite type which, upon detonation, creates a force capable of gaining greater purchase on the burden. Observation of the rock production from the use of straight nitroglycerine, gelatin, and ammonium nitrate explosives is said to have proved that their force is too violent. Most formations actually quarried disclosed the presence of highly resilient rock below the plane of weathering. The disruption of this more resilient rock demands the use of a force which will take hold and stress the rock slowly but surely beyond its elastic limit. This is the kind of force that the new explosive is especially developed to give. Ideal fragmentation, controlled throw of débris, limited back break, and more tons of rock per pound of powder used are said to be secured.

Dragline Caterpillar Mounting Improved for Soft Ground

A new oversize caterpillar mounting has been developed by Bucyrus-Erie Company, South Milwaukee, Wis., for application to that company's 52-B dragline. It is extra long (21 ft. 2 in.), providing greater stability when on soft bottom. It is extra wide (13 ft. 6 in.) and can be furnished with 36- or 42-in. wide links to give greater bearing area than the standard mounting used for solid bottom. The other essential requirement of liberal clearance under the truck frame has been met, and mud guards protect the simple driving machinery. The 52-B dragline handles a 1½-cu-yd. heavy-type bucket on a 75-ft. boom.

For Washing Gold Gravels

A mechanical gold pan for either hand or power operation, and which can be used as a concentrator or for clean-up purposes, has been developed by J. F. Geary, of the Galigher Company, Salt Lake City, Utah. The standard size (12x36 in.), hand-operated, will handle 1 cu.yd. per hour. Equipped for hand-operation, it weighs only 165 lb. Its legs can be folded so that it measures only 50x22x18 in. It requires little water and will work automatically on gravel up to 3 in. The head motion is novel.

Motor and Speed Reducer Combined

Combining motor and speed reducer in a single unit, the so-called "Motoreducer" has been developed by the Falk Company, Milwaukee, Wis., in three styles—namely, "Integral," "Flexible," and "All Motor," depending on the nature of the combination. All three styles may be had in horsepower from ¾ to 75, with ratios from 4.6 to 288. Non-planetary, helical gears of special make are used. Minimum efficiency is 97 per cent.

Methods for Safe Electrical Installation Defined—A New A.S.A. Standard

A new mining standard (M24-1932) has recently been approved by the American Standards Association, 29 West 39th St., New York. Its title is "American Recommended Practice for Safety Rules for Installing and Using Electrical Equipment in Metal Mines." It was developed by a sectional committee under the sponsorship of the American Mining Congress with F. L. Stone, of General Electric, as chairman. Several engineers of the U. S. Bureau of Mines collaborated in its preparation, though not on the committee.

The purpose of the rules is indicated in the title. In preparing them, five basic measures for safeguarding the use of electricity in mines were followed:

1. Remove the contributory causes of accidents or danger.

2. Remove from the vicinity of electrical apparatus all elements that are known to be susceptible to the influence of electricity.

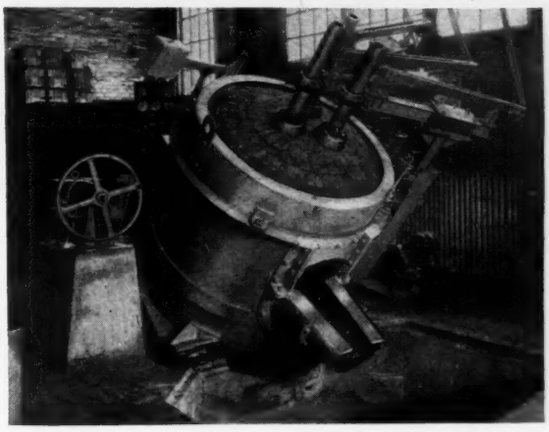
3. Keep the electric current where it belongs, if possible; if not, limit the area by protective devices.

4. Use a large factor of safety in the selection, installation, and inspection of equipment.

5. Have full control of the operation of electrically driven machines.

The rules are divided into four parts: I, General, 21 rules; II, Stationary Electrical Equipment, Buildings, and Enclosures, 20 rules; III, Portable Electrical Equipment Circuits and Conductors, 28 rules; and IV, Circuits and Conductors, 23 rules.

The bulletin has been issued in mimeograph form for the present.



Heroult-type 3-phase electric furnace, intended primarily to be connected to transformers available with larger units. It can also be run as an individual furnace.

Meeting Power Demand Difficulties With an Electric Furnace

When output falls, the power demand charges for large electric furnaces become excessive, so that by installing a smaller furnace an operator may completely amortize the cost in less than a year by the saving in demand charges. A small unit is also more suitable than a large one for many purposes. The accompanying photographic reproduction shows a Heroult-type three-phase furnace designed by Samuel Arnold 3d, consulting engineer, of Pittsburgh, Pa., primarily to be connected to transformers available with larger units, although it can operate as an individual furnace. It can be lined with either acid or basic refractories. At periods of low production the power demand charge will be only one-half to one-fourth of that of a larger unit. The furnace is completely equipped for automatic power control, each electrode having for moving it a separate electric motor acting through spur gears and a De Laval worm gear. In this unit, the worm shafts are extended for the attachment of hand wheels for manual operation, when required. The furnace is also tilted by means of a De Laval worm gear operating a crank. In the cut, the hand wheel is shown installed upon the worm shaft, which is supported by outboard bearings. The top of the gear casing provides for the mounting of an electric motor, which would be connected to the worm shaft by spur gears, as is done in the case of the electrode motors.

Anti-Friction Bearings for Mine Cars

Painful experience of mine executives with experimental inclosed ball bearings of early design is given as the reason why anti-friction bearings have been adopted slowly for mine cars or trucks in certain districts. Robert Hudson, Ltd., Leeds, England, say that, after many failures in the past, they have evolved three types of ball and roller bearings which can be fitted with standard races made by all the leading makers of ball and roller bearings. These are known as the Cabo, Bako, and Faro types, respectively. The first is for cars

having inside bearings and fast wheels; the second for cars or wagons with inside bearings and differential wheels, suitable for sharp curves; and the third for cars having outside bearings. All are practically unbreakable, being made of high-tensile steel, and will give service for a long period without lubrication or attention.

Diesels to Run on Oil or Gas

Covering power requirements from 50 to 1,000 hp., in 2- to 8-cylinder units, an improved line of four-cycle direct-injection moderate-speed diesel engines has recently been introduced by the Worthington Pump & Machinery Corporation, Harrison, N. J. These diesels can easily be converted to run on manufactured or natural gas.

Nevada's New Car Dumper

A 40-ft. three-hook type clamp, revolving car dumper has recently been built by the Wellman Engineering Company, Cleveland, Ohio, at the McGill plant of Nevada Consolidated Copper. It handles cars ranging in size from 9 ft. 8 in. to 11 ft. high, 9 ft. to 10 ft. 9 in. wide, up to 38 ft. in length over couplers, the heaviest of which will not exceed a weight of 50,000 lb. for empty cars and 100 net tons of ore plus approximately 10 per cent overload, at a maximum rate of 30 cars per hour.

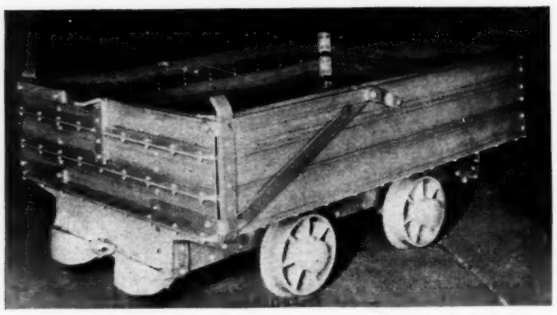
Rubber Transmission Belts Endless

A new rubber transmission belt and a simple outfit for making it endless have been introduced by B. F. Goodrich Rubber Company, Akron, Ohio. The belting, known as "Highflex Junior," is built up of a large number of plies of specially woven fabric designed primarily to give the characteristics necessary when transmission belting is to be spliced endless directly on the machine. It is made in 6 in. widths and under.

For V-Belt Drive

A new line of single and multiple V-groove pulleys for use with its cog-belt drives, for ratings up to 7½ hp., has been introduced by Dayton Rubber Manufacturing Company, Dayton, Ohio. The new pulleys are of heavy gauge pressed steel welded at rim and web.

Aluminum Alloy in Mine Car



All parts of this mine car except the bumpers, axle boxes and wheels are made of strong aluminum - alloy structural shapes, bar, or plates. By thus substituting light alloys, a saving in weight of 1,570 lb., or 46 per cent, was made by the Duncan Foundry & Machine Works, Inc., of Alton, Ill., which built the car for service at East St. Louis.

Cableways for Quarries and Open Pits

A line of tautline cableways of improved design and construction has been developed by Sauerman Bros., Inc. Two types are included: light duty, a 1- or 2-ton size for spans up to 500 ft.; heavy duty, a wide range of sizes for loads up to 30 tons. In the former the carriage is of the two-wheel type. The latter may be had with 3-, 4-, 6-, or 8-wheel carriers, according to capacity. Design of the steel towers has been changed radically.

Another Corrosion Resisting Steel

A new two-ply stainless steel that may be deep-drawn, stamped, welded, formed, or polished has been developed by Ingersoll Steel & Disc Company, Chicago, a division of the Borg-Warner Corporation. A perfect bond is said to have been obtained between the stainless steel and the carbon steel back. The product is said to be low in cost and able to resist corrosion of mine waters and acids.

Small Separator for Batch Work

For laboratory and industrial work on a small scale, the Raymond Bros. Impact Pulverizer Company, Chicago, has developed a compact air separator having a capacity ranging from 100 to 700 lb. per hour, which duplicates the larger mechanical air separators made by the company.

INDUSTRIAL NOTES

General Refractories Co., Philadelphia, Pa., has made arrangements whereby it will market Carbex (silicon carbide) commodities manufactured by McLeod & Henry Co., Troy, N. Y.

Diesel Engine Manufacturers' Association recently elected H. Birchard Taylor, of Philadelphia, president, succeeding George W. Codrington, of Cleveland, who was elected chairman of the board. E. T. Fishwick, of the Worthington Pump & Machinery Corporation, continues as chairman of the executive committee.

BULLETINS

Conveyors. Redler Conveyor Co., 117 Liberty St., New York. Redler "en masse" conveying, for various materials. Pp. 20.

Safety Footwear. Mine Safety Appliances Co., Pittsburgh, Pa. A new line of safety shoes and pacs. Pp. 4.

Crushers. Patterson Foundry & Machine Co., East Liverpool, Ohio. The "Gyro-Centric" crusher for non-metallic and metallic minerals and other materials. Pp. 8.

Refractories. Quigley Company, 56 West 45th St., New York. "Cast-Refract," a dry powder refractory which can be mixed with water on the job to make any desired shape. Pp. 4.

Excavating Equipment. Bucyrus-Erie Co., South Milwaukee, Wis. Bulletin. "Red Arch" dragline bucket.

Rock Drills. Sullivan Machinery Co., Chicago, Ill. Loose leaf catalog. Equipment, parts and supplies used with hammer drills.

Roasters. Bethlehem Foundry & Machine Co., Bethlehem, Pa. Bulletin. Wedge roasters for metallurgical work, clay burning, filter medium regeneration, and byproduct recovery.

Boiler Plant Equipment. Northern Equipment Co., Erie, Pa. Leaflet. Type DA Copes feed-water regulator.

Underground Haulage. Greenwood & Batley, Ltd., Albion Works, Leeds, England. Catalog. Greenbat electric locomotives.

Magnetic Separation. Dings Magnetic Separator Co., Milwaukee, Wis. Bulletin. Type IR super-high intensity magnetic separator.

Blowers, Compressors. Roots-Connersville-Wilbraham, Connersville, Ind. Bulletin 170-B10. Centrifugal blowers and compressors, single and multi-stage for handling air or gas under pressure or vacuum.

Haulage Material. Westinghouse Electric & Manufacturing Co., East Pittsburgh, Pa. Catalog 234. Trolley line material and rail bonds. Pp. 100.

Hoists. General Electric Co., Schenectady, N. Y. Catalog GEA-1593. Electric equipment for mine hoists. Pp. 23.

Crushers. Traylor Engineering & Manufacturing Co., Allentown, Pa. Bulletin 111. Pp. 10.

Explosives. Ensign-Bickford Co., Simsbury, Conn. Cordeau-Bickford detonating fuse. "A textbook of the new technique in blasting." Pp. 38.

Placer Mining. Mine and Smelter Supply Co., Denver, Colo. "Shades of 49," a 12-page catalog of portable placer machinery and supplies.

Mine Conveyors. Jeffrey Manufacturing Co., Columbus, Ohio. Bulletin 546. Belt type conveyors for mine service. Pp. 8.

Valves. The Lunkenheimer Co., Cincinnati, Ohio. Leaflet. "Slaset" lubricated rotor valves. Pp. 4.

Crushing. The Hardinge Co., York, Pa. Bulletin. The Hadsel mill, a report by R. G. Hall. Pp. 8.

Rock Drills. Ingersoll-Rand Co., 11 Broadway, New York. Bulletin, Form 4101. Rock drills and sharpeners. Pp. 34.

NEWS OF THE INDUSTRY

WASHINGTON, D. C.

Gold Reserves Appreciate. Improvement in the status of the gold standard, and an increase in the amount of the metal held in the United States, is reported by the Federal Reserve Board. As compared with the middle of 1928, it stated, the central gold reserves of the principal countries of the world outside the United States had increased about $1,800,000,000, or 30 per cent, because of a larger output of gold mines and the release of gold from private holdings in several countries, particularly China, India and Russia. . . . At a meeting of the Industrial Health Section of the National Safety Congress on Oct. 4 the subject of the dust hazard in industry was discussed. Dr. L. U. Gardner explained the action of various dusts on lung tissues. Atmospheric dust contents in industrial locations was discussed by J. J. Bloomfield, of the U. S. Public Health Service. Prof. T. Hatch, of the Harvard School of Public Health, a recent contributor to *Engineering and Mining Journal* on the subject, outlined methods of dust control.

CALIFORNIA

Low-Grade Ore Development. Carson Hill Gold Mining has reopened the Carson Hill mine in Calaveras County, and intends to exploit the large reserves of low-grade ore said to exist along 3 miles of the Mother Lode. The deposit lends itself to cheap mining methods, and a mill is available with a capacity for 700 tons per day, which is to be reconditioned. . . . Idaho-Maryland, operating at Grass Valley, has added concentrating tables and dispensed with amalgamating plant. The introduction of flotation as a supplementary process is expected to reduce losses in the tailing. Ore reserves have been materially increased by developments on the 1,000-ft. level. . . . Sierra del Oro Mining, operating the North Fork mine, in Sierra County, has developed sufficient ore to justify the erection of a mill, arrangements for which are being made. Yellow Jacket Consolidated, also in Sierra County, is developing the Yellow Jacket and Osceola groups. . . . The A. S. & R. company is reported to have abandoned work at the Big Blue mine at Kernville. . . . Beebe Mining, operating in the Georgetown district, near Placerville, is changing the system of mining from glory-hole to shrinkage stoping. Total costs of mining and milling have been about $1 per ton, with an output of 175 tons per day. . . . Ainsworth Consolidated Mining, owners of the Gold Corral placer properties near Yreka, Siskiyou County, have acquired a complete plant to recover gold. Operations are expected to commence in the near future. . . . . A company has been formed to renew efforts to recover gold from the Mammoth Channel, in Butte County. Surveys have been made that are said to have demonstrated the practicability of operating by hydraulic and sluicing methods, with prospect of profitable result.

Amalgamation and flotation is to be used on a small plant now being erected at the Hidden Treasure mine, near Tuolumne, for the recovery of gold from high-grade ore.

Texacala Mining Company, West Point, Calif., is installing a 165-hp. Busch-Sulzer diesel engine which is direct-connected to a 120-kw. generator. The flotation cells the company intends using are of the Minerals Separation design and are now being assembled by the Joshua Hendy Iron Works, San Francisco. The company is building a 50-ton flotation mill at the Black Wonder mine. This mill will be in operation, it is expected, within the next 30 days. The Texacala company was recently organized. Ed. S. W. Bateman, of Fort Worth, Tex., is president, and F. E. Browne is manager.

ARIZONA

Ohio Capitalists Invest. The Jaeger prospect, in the Comababi Mountains near Tucson, has been purchased by Akron Gold Mining, which plans to deepen the shaft to 500 ft. Akron Gold Mining is a closed corporation with no stock for sale. E. R. Rice is in charge of operations. . . . United Verde Extension has obtained an option on the controlling interest in Leviathan Sulphur, developing a sulphur deposit near Markleeville, Alpine County, Calif. . . . United States Marble, of Denver, has purchased the onyx quarry near Mayer and has commenced operations.

Hayden Smelter to Operate. Work at the Magma mine and mill was resumed on Sept. 16. Between 300 and 350 of the company's old employees were given half-time employment. The Hayden smelter of A. S. & R. will be reopened for about 30 days on Nov. 1, at which time the Nevada Consolidated mill at Hayden will have produced enough concentrates to run the smelter a month. Several of Arizona's placer and lode districts are being worked for gold, especially by small operators. The Central Arizona Mining Association and the Yavapai County Chamber of Commerce, both with headquarters at Prescott, are well informed about placer mining conditions and will supply information to anyone who inquires. . . . A Pasadena group, headed by R. L. Johnston and R. S. Witcher, has obtained the Big Jim mine at Oatman. . . . R. W. Smith, referee in bankruptcy for the Katherine Gold Mining, has granted the application of the Thornton Engineering Corporation, of Los Angeles, for an extension of time for the payment on the property. The corporation is operating the Katherine mill on ore from the Katherine and Frisco mines, and on custom ore, and hopes to be able to put the Katherine on a profitable basis. Ore from the Roadside mine, which Ross Householder has under lease, is being hauled to the Katherine mill in 10-ton trucks at the rate of 100 tons per day. . . . A 50-ton mill for treating ore from the old Harrisburg mine, 6 miles south of Salome, is being erected by Harrisburg Milling. Amalgamation and cyanide will be used, and the mill is to be in operation by Oct. 15. The plant will be operated on a custom basis, and several near-by properties will be benefited. . . . The M. A. Hanna company, of Cleveland, has purchased the property of United States Copper Mining at a sheriff's sale. The holdings sold consist of 20 patented mining claims, 60 unpatented claims, and 39 mill sites near Metcalf, Ariz. The patented claims comprise the Lesinsky group, from which copper ore was shipped many years ago. M. A. Hanna company was engaged in diamond drilling the property in 1931, but stopped operations there early in 1932.

One of the furnaces at the C. & A. smelter at Douglas has been put into commission.

PENNSYLVANIA

Rare Kaolin Mineral Found. A 6-in. seam of dickite, a hydrated aluminum silicate of the kaolin group, was recently found in a 10-ft. conglomerate bed beneath an anthracite vein by miners who were driving a rock tunnel in the Pine Knot colliery of the Philadelphia & Reading Coal & Iron Company, near Pottsville, Pa. The mineral, previously reported from only ten localities in the world, is in the form of a glistening white powder of monoclinic crystallization. The identification, first made by the State Geological Survey and the Pennsylvania State College staff, has been verified by the U. S. Geological Survey.

NEVADA

New Equipment Pays for Itself. The new installations at the crushing plant at McGill of Nevada Consolidated, costing $283,000, will effect a saving of 4c. a ton. If production is maintained at same rate as last quarter, the plant is expected to pay for itself in about two years. The plant handles pieces of ore of any size that can be loaded by the shovels at Ruth. Secondary blasting of ore at the mine is avoided, saving three cents a ton. . . . The latest Nevada gold rush is to Clarkedale, on the north-south highway, 70 miles south from Tonopah, or 30 miles north of Beatty. John Clark, the discoverer, has sampled a rhyolite vein, cropping for 300 ft., reporting assays of $6 to over $100 per ton. . . . Mine owners and leasers of Boulder, Colo., have organized a cooperative milling company, securing a mill unit that is being equipped to handle 100 tons of ore per day. With this mill the mine owners and leasers hope to secure a profit from low-grade ore now being placed on the dumps.

Active Interest in Gold. The old Shenandoah mine, in the Yellow Pine district, has been sold. The property, 8 miles west of Goodsprings, was located 50 years ago and has produced lead and silver. The ore is now found to carry molybdenite. A 50-ton shipment of concentrate is now on the way to Belgium, where it will be treated. If the project proves successful, the mine will be developed during the coming winter and a concentrating plant erected in the spring. . . . The Rice & Adams placer property, 8 miles north of Yerington, is expected to be in production early in October. A steam shovel is on the ground, also material for sluice boxes and trommel screens. A storage tank has been installed. The capacity of the shovel is about 800 yards a day. The new mining camps of Nevada are making a good showing during the past month. Scossa has proven good grade of milling ore on the 400-ft. level of Dawes Gold. Work has started on several properties. Payment has been made on the Overhang claim, and a company incorporated to work the property. Scossa Homestake Mining has been incorporated to operate 40 acres of patented ground, purchased from the Southern Pacific, with a good surface showing. . . . Storm Cloud, at Eddyville, near Mina, has made its tenth weekly shipment of high grade. Heretofore these shipments have averaged over $1,150 per ton. Clarkedale Gold Mines, at Clarkedale, has been incorporated, to operate the discovery ground. Eleven leases have been let. C. D. Wilkinson is consulting engineer. . . . Another new strike is reported at Palmatoro, in the Palmetto Mountains 16 miles southeast of Silver Peak. . . . George Wingfield reports that crosscutting on the 300-ft. level of Goldfield Con., to undercut the depth limit in the Fitzgerald-Svorcan lease, was started Sept. 21. Ore from the lease is now going out at the rate of 50 tons per week. Assays average $60 per ton. Sixteen leasers are working on the Goldfield Con. estate and twelve on the Florence. . . . After clearing the company of litigation, Frank Horton has started sinking the main shaft on the Horton Weepah, formerly Electric Gold, and on which estate a rich discovery a few years ago started the Weepah boom. . . . The cleanup for last half of August at the Bradshaw, Inc., was $15,947.

COPPER COUNTRY

Concentration Research Progresses. In research work on concentration of low-grade iron ore, of which there are thousands of millions of tons in upper Michigan, the geological and metallurgical departments of the Michigan College of Mining and Technology, at Houghton, are concentrating efforts on material from the Gogebic range. Coarse-banded iron rocks of that district offer favorable samples for experimental work. T. M. Broderick, chief geologist for Calumet & Hecla Consolidated, is in charge of the field work, and A. T. Sweet, head of the metallurgy department of Michigan Tech., is conducting the experiments. Concentration is being effected by use of a dense medium in a jigging operation.

A total of 6,244,000 lb. of copper was shipped out of the Michigan district in August, most of the metal being delivered by Calumet & Hecla and Copper Range. A shipment of 1,450 tons of Danish flint pebbles, to be used in the regrinding mills, was received by Calumet & Hecla by boat direct from Denmark. Pebbles from various parts of this country and others have been tried, but the Danish product has proved the most durable and efficient. . . . Calumet & Hecla No. 21 shafthouse, Kearsarge lode branch, was destroyed by fire on Sept. 5. Operations were suspended at No. 21 in 1913, since which time the shafthouse had not been used. . . . Prof. A. T. Sweet, head of the metallurgy department of the Michigan College of Mining & Technology at Houghton, who has been appointed a member of the Michigan project committee and chairman in the Upper Peninsula of Michigan for the National Committee for Trade Recovery, will list and analyze all regional projects in line to receive loans from the government. The primary object is to provide productive employment, develop purchasing power, and revive industry by construction of deferred but essential public works. Professor Sweet will be assisted by project chairmen and county chairmen whom he will appoint. Construction costs, he says, are now so low as to warrant greatly increased activity, especially when so many unemployed men must have productive work or financial aid. The national committee hopes to convince public officials that a difference exists between economy in current operations and false economy in deferring capital investments for basic necessities.

Michigan Copper Looks Ahead. Although the copper tariff has been without effect, because of the large surplus of metal, it has served to hearten the mining companies and the people of the Michigan district. The feeling prevails that, with the copper duty now in effect, the industry will revive with a return of improved conditions in business generally and absorption of the surplus.

UTAH

Employment Plan Proposed. Incorporation of Utah Transportation & Development by representatives of mining companies in the Bingham district and managers of the three Utah smelting companies is being planned, preparatory to applying for a $2,500,000 loan from the Reconstruction Finance Corporation to drive a 43,000-ft. drainage and transportation tunnel. Of this, $2,000,000 will be spent to drive the tunnel, to accommodate two rail lines and a water conduit. The remaining $500,000 will be used to finance the moving of 2,400 people from the congested district of Carr Fork, Bingham Canyon, and the building of homes in Tooele County. The plan aims to provide flexible transportation of ores from Bingham mines to smelters in Salt Lake and Tooele County; ventilation and drainage for mines of the district, most of which are pumping and hoisting ore from great depths; better living quarters for employees. It will supply employment for 500 to 700 men for eighteen months to two years. . . . Howard Fitch, Tintic, is developing a placer property in the East Fork of the Beaver Dam Wash, Washington County, Utah. Test shafts are being dug and a portable M. & S. placer machine is being used. . . . A fire in Bingham on Sept. 8, wiped out the central section of the Highland Boy district in Carr Fork.

IRON COUNTRY

Concentrate Is Being Stockpiled. An unusual sight can be observed on the Mesabi range—four washing plants running full time, but no concentrate being moved to Duluth for shipment. It is being stockpiled because work is needed to carry employees through the winter without suffering, the companies wisely preferring to supply work rather than to spend the money for charity. If ore is removed from certain parts of a pit the company can lighten the tax burden by returning the lease to owners, and let them carry the no-profit load until times improve. Some operators feel that the state may remove the stockpile tax for next season, to encourage underground mines to open. By cutting wages another 20 per cent, ore is being placed on the stockpile at a cost lower than can be secured next season if times improve. This pit activity will continue until the freeze-up. Washing plants now working are the Danube mine, at Bovey, operated by Pickands Mather; the Patrick mine, at Cooley; the Harrison mine, at Nashwauk; and the Margaret mine, at Buhl—all operated by Butler Brothers. About $750,000 tons of concentrate will result from the aforementioned operations.

Electric Shovels Displace Steam Units. The opinion of the Oliver Iron Mining as to the economy of electric over steam shovels is shown by the number of steam shovels that have been shipped as scrap during the summer. Nine units of the 4-yd. railroad type have been shipped, and the crew is still busy wrecking additional shovels. The total may amount to over 20 before the work is completed. With only electric shovels operating in the Hull-Rust pit, the time when complete electrification will be accomplished seems near.

PACIFIC NORTHWEST

Alaska Juneau Profits. Operations in the vicinity of Nome are on a full-time basis, exports of gold dust and bullion in August amounting to $959,359. Work is progressing on a new $100,000 apartment house at Juneau. . . . In August, Alaska Juneau mined and trammed 350,300 tons to the mill at a cost of 23.55c. per ton. Milling cost 18.41c. per ton. Total costs aggregated $159,500, and operating profit was $100,500. The surplus stands at $101,700. . . . Twisp Mining & Smelting, of Yakima, Wash., owner of the Alder mine, at Twisp, Okanogan County, plans to bring the mine to production and to install a 250-ton plant. . . . Dome Mines, of Porcupine, Ont., Canada, is reported to have acquired a prospective gold property, the Black Pine, near Elk City, Idaho. . . . A new gold mill at Golden and another between Oro Grande and Elk City, Idaho, are expected to be in operation soon.

MONTANA

New Flotation Plant Operates. William H. Emmons, head of the geological department of the University of Minnesota, recently examined the ground of Washington Bar Gold Mining at Nomo, Madison County. . . . The Boss Tweed & Clipper is now operating its flotation plant at Pony, Madison County, on a three-shift basis. Ore is delivered from the tunnel to a storage bin, from which it is sent through a crusher, grizzly, and elevator to the crushed-ore bin. From there the ore is put through a ball mill, in circuit with two Dorr classifiers, the overflow from which is sent to a sludge tank and through Fahrenwald flotation machines. Final concentrate is thickened and dewatered on an American filter. The plant has a capacity of about 125 tons, the ore runs from $7 to $12 per ton in gold, and the mill is concentrating at a ratio of about 15 to 1. Electric power, from the lines of Montana Power, is being used, and electric haulage underground will probably be adopted. R. E. Sawyer is general manager; Charles Drackert, mine superintendent; and Tobey Peterson, mill superintendent. Iron Rod Gold Mines has been taken over by Golden Gate Mining. The property is about 37 miles from Butte, on the Vigilante trail. The ore was formerly treated in one of the first cyanide plants built in Montana. The present operators plan to drive a tunnel to cut the main veins at two to four hundred feet below the old workings, thereby avoiding the hoisting of ore and pumping of water. A gasoline driven compressor has been installed, and a line will connect with the line of Montana Power. L. O. Thomas, mine operator of the Coeur d'Alenes, is president and general manager, with headquarters in the Phoenix Block, Butte. . . . The Winnetka group of claims in the Virginia City district, Madison County, has reverted to its owners and is being operated under the supervision of Harold Gohn. The ore, which assays about $15 per ton in gold, is being treated with a Lane mill, plates, and Wilfley tables. . . . Jardine Mining has started operations at Jardine, Park County. Both mine and mill are operating, producing and reducing an ore containing gold, arsenic, and tungsten. When arsenic was high, this property was one of the largest producers of white arsenic in the west, having its own roasting plant. The last car of tungsten sold netted the company close to $25,000. The company has its own hydro-electric plant, one of the largest independent plants in Montana. Harry Bacorn is general manager.

TRI-STATE DISTRICT

New Metallurgical Plants. The new $300,000 mill of Eagle Picher Mining & Smelting is nearing completion. Steel construction work has been finished and equipment is being placed in position. An all-steel conveyor is to be used for the transport of tailing, much of which will be used for mine filling, and some disposed of for ballast and other commercial purposes. . . . A new tailing plant is being placed in operation. Material for the plant will be excavated by electric shovel, transported in 6-ton cars. Concentrating equipment comprises jigs, rolls, tables, and classifiers, flotation machines, with settling and filtering units. The plant has a capacity of 100 tons of tailing per hour. . . . Mines in the Tri-State region now have about 66,000 tons of concentrate on hand, a reduction of nearly 25,000 tons during the past three months. The smelters are carrying no stocks of concentrate, and an optimistic view is taken of the general situation in the region. . . . St. Louis Mining & Smelting recently commenced to ship 15,000 tons of zinc concentrate to the East St. Louis smelter of American Zinc, Lead & Smelting.

NORTH CAROLINA

Mica Industry Shows Strength. Shipments of most classes of mica from the Spruce Pine district have increased remarkably since early in August. More scrap mica has gone from there into the hands of consumers in the last few weeks than during any similar period in several years. Prices of scrap have also been raised nearly 30 per cent from the low levels that have held for so long. Only limited stocks are available locally, and mica is being mined for the first time in three years or more. . . . Dry-ground mica schist is moving as well as at any time since this product was introduced. Fine mica obtained as a byproduct in the refining of china clay is also in steady demand. In fact, local stocks of ground schist and byproduct mica are nil. . . . Prepared roofing manufacturers, who use most of the dry-ground schist and byproduct mica that is produced, are almost entirely without stocks of finished goods. Distributors' and retailers' stocks of roofing are also low. Sales of this kind of roofing have been negligible for some time, and replacements cannot be deferred much longer. Demand for the grades of mica used in producing it is likely to be sustained. . . . Punch and sheet mica are in fair demand, with output encouraging. Sheet-mica processing plants are operating at capacities above those prevailing for some time. Much forward business is in sight, with all orders requiring prompt attention.

CANADA

Quebec Gold Output Stimulated. Results of development are accumulating rapidly in the gold belt of northwestern Quebec, and new producers will be added in quick succession. . . . Work at the Beattie property, 20 miles to the north of Noranda, is concentrated on developing the orebody on the first two levels. The values are reported to be $4 or better, which shows a substantial increase over the average calculated from diamond drilling. W. G. Hubler, responsible for the Amulet and Aldermac flotation plants, has been engaged to erect the first 800-ton unit of the Beattie mill.

Eldorado Gold Mines announces that the plant to treat the pitchblende from Great Bear Lake will be at Port Hope, on Lake Ontario, 60 miles east of Toronto, where facilities for supplies by rail and water are excellent. The first shipment of about 20 tons of radium ore has reached Port Hope.

Mining Activity in Ontario. Wright-Hargreaves is now one of the big mines of Kirkland Lake, the ore in its lower levels down to 3,000 ft. being comparable in size and grade to the ores of Lake Shore. The company will deepen the mine to 4,000 ft., and increase capacity of the mill to 1,000 tons. The Lake Shore report for the year ended June 30 indicates a yield from 834,434 tons of ore of a little less than $15 a ton, the amount being increased from $13 at the beginning of the fiscal year to $17 at the end. Operating cost is still rather high, at $5.24 per ton. Profits were $7,797,011 after all charges, and dividends of $6,000,000 were paid during the fiscal year.

Falconbridge Nickel Expands. Falconbridge Nickel plans an expansion of output. Foundations are being laid for an extension of the present mill. The mine is prepared for an output considerably greater than its present 350 tons a day, and smelter capacity can be doubled by adding a little equipment on foundations already prepared. Additional refinery capacity will be required to meet a larger mine and smelter output.

BRITISH COLUMBIA

Reopening Plans Widespread. Additions have been made to the Pioneer mill, in the Bridge River district, by which capacity has been trebled. A new shaft headframe has been completed and a double-drum electric hoist installed and housed. A 500-ton ore bunker has been constructed at the new shaft-head and has been connected by a short rope tramway to the mill. Underground developments have been satisfactory. The ore going through the mill averages $20 per ton. About 95 per cent of the gold is recovered by cyanide. The present shaft is to be sunk another 500 ft. . . . The Velvet mine, in the Rossland district, is being reopened by Seattle interests. A considerable reserve of low-grade copper gold ore is expected to pay for treatment. . . . Shipments to the Tacoma smelter of concentrate from the Dividend-Lakeview gold mine, near Osoyoos, are being made. The mill has been reconditioned. . . . Considerable activity is noticeable in placer mining in many parts of British Columbia, and several discoveries of importance have been reported. . . . Reopening of the Engineer mine, in connection with a plan for the reorganization of Engineer Gold Mines, Ltd., is projected. The property has had a chequered career, and was extensively explored and developed in 1924 and 1925, when 4,000 ft. of tunneling in addition to diamond drilling was undertaken. The ore occurrence is a large pyritized shear-zone containing gold in quartz stringers, but no substantial reserves have been created. During the last year of mining operations a production was made of 1,814 oz. gold from 1,700 tons of ore milled.

Several New Plants Proposed. Kitsault Eagle Mines is contemplating the installation of a flotation plant on its Skidgate-Southeaster property on Graham Island of the Queen Charlotte group. The company has been engaged in surface and underground exploration here for the past two years, and has succeeded in proving continuity of the ore zone for about 1,200 ft. At the close of 1931 encouraging results were reported from crosscuts on the 100-ft. level of the main shaft; subsequent developments have not only solved the structural problem but given assurance of an ore supply that may justify provision of a mill. Decision will depend on the report of W. G. Norrie-Lowenthal, the company's consulting engineer. . . . The Nickel Plate mine of Hedley Mining in the Similkameen district, closed last year after a gold production in thirty years of over $12,000,000, was recently acquired under option by New York interests on the presumption that new orebodies might be discovered by intensive exploration.

AFRICA

Roan Reduces Smelting Costs. That advantage may be gained by adjusting equipment to tonnage, when tonnage cannot be adjusted to equipment, has been demonstrated by Roan Antelope Copper. The June quarterly report discloses that advantage was taken of a shutdown for an arch repair at the end of May to shorten the furnace and make other alterations to adapt it more nearly to the small tonnage of charge imposed by the restricted production quota. The advantage of the change is shown by the cost of smelting per long ton, £2.72 in July, against £4.25 in April. Thus by adapting the furnace to the tonnage Roan Antelope has cut about a quarter of a cent per pound off the cost of producing copper. Despite restriction of output and the necessity of making the small production carry the total overhead, Roan's costs are now under 4c. per pound of blister copper delivered to consumers in Europe.

Roan Antelope Plant Capacity. Operating results have demonstrated that the Roan Antelope plant has an annual output capacity of 200,000,000 lb. of copper. In designing the concentrator the original idea was a capacity of 100,000,000 lb. a year, but this was raised, by additions, to 120,000,000 lb. before construction was completed. When operations began, both ore and plant acted better than had been expected, and it was soon evident that capacity would be at least 150,000,000 lb. Now an annual capacity of 200,000,000 lb. is understood to have been proved beyond question. The importance of this is not confined to the period when curtailment of output will no longer be necessary; it also signifies that a much smaller expenditure than estimated will be required to provide plant for the planned future increases of productive capacity.

Rand Results Encouraging. Plans for the developing of New Lease Areas in the South-East Rand are developing satisfactorily. Excellent results are being obtained at depth on West Rand Consolidated and Luipaardsvlei Estates. The outlook at the City Deep is improving. Sub Nigel profits show steady advance, and development results are satisfactory. . . . The South African diamond industry shows signs of recovery. A more hopeful feeling is in evidence at the tin fields and copper mines. The Main Reef has been intersected in the west sub-incline shaft of West Rand Consolidated at the horizon of Level 31, and assays for the first 50 ft. average 14.6 dwt. over 68 in. This is the lowest point of development in the western section of the mine. . . .

Labor Conditions in Belgian Congo. A commission of inquiry, financed by Carnegie Foundation, and composed of American, British, and South African university professors—no Belgian citizen being a member—visited Katanga recently to study labor conditions. J. Merle David, director of the Department of Social and Industrial Research and counsel of the League of Nations, was at the head of the commission, which was favorably impressed at the manner in which the natives are being treated and their interests considered.

MEXICO

Interest in Placer Mining. A concentration plant has been completed by Cia. Minera de Campo Morado near Tlacotepec, Guerrero. The company is controlled by Oriental Copper. . . . A 500-ton gravel washing plant has been shipped from the United States to its holdings along the Rio Yecorato, Yecorato municipality, Sinaloa, by the Cia. Placeras de Oro de Yecorato. Electric power will be supplied by wood-fueled steam engine and generator. . . . Officials of Moctezuma Copper deny reports published by Mexican newspapers that the company has sold its properties at Pilares, Sonora. . . . An option on the Los Azules gold mine, near Santa Barbara, Chihuahua, has been taken by Negociacion Minera de San Rafael y Anexas. Sampling is to begin soon and erection of a mill is being considered. . . . Blast-furnace operations have been resumed by Monterrey Iron & Steel, at Monterrey, Nuevo Leon.

Mexico Restores Standard, Encourages Gold Mining. Assisting prospectors and small-scale operators by placing unexploited mineral lands in the hands of cooperative societies of mine workers is being considered by the Ministry of Industry, Commerce & Labor. The Ministry has ordered its agents and the federal labor inspectors to report details of unoccupied mineral lands in their jurisdiction, and of properties that could be worked by cooperative units in accordance with the owners' desires and acceptances. The Ministry will arrange with the authorities of zones in which such properties are located for aiding cooperative units to work them. The Ministry has asked the Ministry of Finance to grant reduced federal taxes to enterprises that will establish treatment plants in the mining regions.

Erratum—Misplacement of the decimal point resulted in an erroneous statement being published in the August issue in regard to the May output of the San Luis Mining Company, shipped to Selby, Calif. The correct figures are as follows: Gold, 107.453 kilos; silver, 6,035.648 kilos.

CHINA

Barter of Mineral for Grain. Foreign sales of tungsten from the Canton district increased considerably during the second quarter of 1932, consular reports indicate, owing to large shipments to Italy. This is said to be the outcome of a contract between Italy and the Soviet Union, whereby the former is to supply 1,000 tons of wolfram ore in exchange for wheat. Confirmation is lacking in regard to a report that Indo-China will exchange zinc for Soviet wheat.

NEW ZEALAND

Placer Exploitation Active. Amalgamated Kawarau Gold Mining is building a dam below the Arrow River, near Dunedin, the intention being to exploit the deposits of gold-bearing material in the Kawarau River, according to U. S. Trade Commissoiner J. B. Foster, Wellington, who also reports that a London syndicate has secured an option on 19 miles of the Molyneaux River, on which two large dredges are to operate.

AUSTRALASIA

Gold Mining Activity Continues. New flotations of Australian gold mines include New Carshalton Gold, formed to acquire a lease at Bendigo; North Hercules Extended, formed to acquire gold properties at Bendigo; Mount Oriental Gold Mining, to acquire a mine at Wandiligong, Vict.; Dividend Gold Mining, to develop an auriferous reef at Poseidon, Vict.; Golden Ridge Option, to prospect a gold-bearing reef in the Woods Point district of Victoria; Lightning Hill Gold Mining, to develop the Lightning Hill mine at Bendigo; Golden Mile Cracow, to acquire leases at Cracow, Queensland; North Kalgurli Central Gold; and New Brown Hill, to prospect and develop leases at Kalgoorlie, W. A. Formation of so large a number of companies demonstrates the activity in gold mining throughout Australia. . . . The Federal Government intends to review the present gold bounty. With a probable Australian gold yield of 900,000 oz. for 1932, the bounty will be £250,000, compared with £85,000 for 1931. The present price of gold is £7/5/- per ounce. . . . An important deposit of the high-grade sillimanite has been located 20 miles west of Broken Hill, N. S. W. The outcrop is 200 ft. long and 20 ft. wide. Analysis shows 93.25 per cent aluminum silicate and a minor amount of ilmenite. A good demand exists in Australia for this type of refractory. . . . After an extended visit to Mount Isa Mines, Queensland, H. A. Guess, vice-president, American Smelting & Refining Company, stated that revenue from the sale of the lead bullion was not sufficient to cover operating expenses, 40 per cent of which are taken by rail and ocean freights. For the twelve months ended June 30, Mount Isa treated 453,997 tons of ore, producing 87,500 tons of lead concentrate assaying 43 per cent lead, which on smelting yielded 38,954 tons of bullion. All the bullion was shipped to the Britannia refinery near London. In June the company treated 49,039 tons, or 6,300 tons more than during the previous month, for 3,940 tons of bullion. The monthly output of bullion is to be raised to 5,000 tons, which is about one-third of the output from the Broken Hill group of lead-zinc mines. . . . At the Mount Coolon gold mine, Queensland, steady progress is being made in the erection of a 200-ton mill, expected to be in operation soon. . . . Myrtleford Molybdenite, Ltd., has been formed to develop a molybdenite mine at Everton, Victoria, idle since 1926. . . . At Rokewood, Victoria, a prospector recently found a 23-oz. nugget near the surface. At Wenderburn, at a depth of 11 ft., a 36-oz. slug was unearthed. . . . Gold production in Papua, formerly British New Guinea, for the year ended June 30, 1931, was 5,918 oz. New Misima recovered 4,699 oz. from 10,879 tons, the other producers being Mount Sisa Goldfields and Ara Creek. Gold was first located in Papua in 1888, since when 514,000 oz. has been won, principally from the Murna, Louisiade, and Yodda divisions. In the adjoining territory of New Guinea, output for the nine months ended March 31, 1932, was 67,747 oz., an increase of 9,900 oz. in the year ended June 30, 1931. This total will be rapidly augmented when the second dredge of Bulolo Gold Dredging and the treatment plant of New Guinea Goldfields commence operations. . . . The treatment plant of New Guinea Goldfields has been completed, and a trial run proved satisfactory. The company expects a profit of £200,000 per annum from ore treatment. Alluvial Tin (Australia), which controls several tin-dredging companies in Malaya, has formed Alluvial Gold, Ltd., to search for gold dredging properties throughout Australia.

SOUTH AND CENTRAL AMERICA

Gold Mining Provides Unemployment Relief. Lampa Mining, owner of a silver-copper deposit near Santa Lucia, between Arequipa and Puno, Peru, is seeking local sources of sulphur, preparatory to the erection of a treatment plant planned to employ crude sulphur in the production of manganese sulphate, in addition to concentrated silver and copper shipping products. Construction of a metallurgical plant has begun. . . . Interest is being shown in the placer gold deposits in the eastern slope of the Andes. Several American miners and prospectors are operating in the district. The only vein mine in large-scale operation is the Santo Domingo, on the headwaters of the Inambari, which continues in normal operation. The Peruvian Government is planning to send expeditions to explore and work the river placer deposits, starting in the Macusani district, a tributary of the Inambari, to the west of Santo Domingo. This idea is being promulgated as a relief measure for men without jobs. The government is also considering modifications of the existing mining laws relative to gold properties, designed to facilitate the entrance of foreign capital into the larger gold-bearing districts. Similar gold prospecting is being carried on in Bolivia.

Placer Mining in Chile. A decree issued by the Government of Chile reserves all sections of the country thought to have deposits of gold-containing sand for the Federal Government, according to a report to the U. S. Commerce Department. These reserved districts may be exploited only by the State or through concessions granted by the State to private interests. This decree annuls previous decrees that have been promulgated on this subject.

Manganese in Ecuador. Report of the existence of large deposits of manganese in Ecuador, at Sacachispa, near Pasaje, 25 km. from Puerto Bolivar, has recently been made to the U. S. Department of Commerce. Analysis of a sample is said to indicate 58.1 per cent manganese, 2.87 silica, 1.07 iron oxide (Fe_2O_3), 0.47 phosphorus pentoxide, 1.30 magnesia, and traces of sulphur and lime. The property is 2,240,000 square meters in extent. Construction of a 7-km. railroad from the mines to Pasaje would be necessary. A narrow-gage road from the latter point to Puerto Bolivar is already in existence. Large steamers can be accommodated at the port.

EUROPE AND ASIATIC RUSSIA

Soviets Find Iron Ore. Immense deposits of iron ore have been discovered in Western Siberia, 125 miles from the iron furnaces at Stalinsk, now producing 2,000 tons of pig iron daily. The discovery will greatly facilitate the development of the industry, the ore lying less than 70 miles from a railroad. . . . The shaft of the Stantrg mine of Trepca Mines, operating in Yugoslavia, has been completed to the deep adit, at 610 meters, and put into service. Development work has started; over 25,000 cubic meters of waste has been quarried on the surface to fill stopes. The 50 per cent addition to the mill is now complete. . . . According to the United States Consul at Frankfort on the Main, plans are being made to construct a zinc refinery at Magdeburg, Germany, for an initial production of 40,000 tons, to be increased later to 60,000 tons, the intention being to avoid imports of electrolytic zinc. If the plant is erected, operation is not likely for two years.

Copper Costs and Plans in Europe. Rhokana Corporation, operating in Northern Rhodesia, is said to be delivering blister copper in Europe at a cost under the lowest price at which standard copper has ever sold—for less than 4c. per pound, despite curtailment of output. By drawing on the rich ores in its N'Changa-West mine, and operating only its present plants at capacity, it could produce as much copper as any one company, except Kennecott, has produced in any one year. . . . Representatives of all producers other than American will meet in Europe soon to plan for continuance of copper production curtailment. Where the conference will be held and what it will do will depend in large measure on whether or not Great Britain carries out the Ottawa plan for a tariff of 2d. per pound on copper . . . . Trepca Mines, operating in Yugoslavia, produced 7,323 tons of zinc concentrate in August assaying 50.1 per cent zinc. This constituted a new high record output.

Demand for Diamonds Increases. Seven thousand diamond cutters are at work at Antwerp, 1,200 having been added in recent weeks. Fine stones are fetching from 8,000 to 9,000 fr. per carat, with demand increasing for diamonds in this class. Stocks of small stones are low, and the market is being stimulated by purchasing activity.

Austrian Lead Smelter Resumes. The Bleiberger Bergwerks-Union company in Klagenfurt, Austria, is operating at Bleiberg-Krueth, and the lead smelter at Gailitz is also in commission.

Germans Seek Low-Cost Beryllium. After three years of experimentation, the German Beryllium Research Society announces the development of a new process for the production of cheaper beryllium, according to the representative of the United States Department of Commerce at Frankfort-on-the-Main. The benefits to be derived by the addition of small amounts of beryllium to other metals is recognized, and opportunity for extensive consumption is expected if the price can be lowered.

Daily and Average Monthly Prices of Metals

September, 1932

United States Market

Sept.	Electrolytic Copper: Domestic Refinery	Electrolytic Copper: Export Refinery	Straits Tin: New York	Lead: New York	Lead: St. Louis	Zinc: St. Louis
1	5.525@5.775	5.600	24.300	3.600	3.450	3.100
2	5.650@5.775	5.600	25.125	3.600	3.450	3.250
3	5.775	5.650	25.250	3.600	3.450	3.250
5	Holiday	5.750	Holiday			
6	5.775@5.900	5.900	25.375	3.600	3.450	3.375
7	5.900@6.025	5.925	25.625	3.600	3.450	3.400
8	6.025	5.900@5.950	25.500	3.600	3.450	3.450@3.500
9	6.025	5.825@5.900	25.200	3.600	3.450	3.450
10	6.025	5.825@5.900	25.250	3.600	3.450	3.450
12	6.025	5.825@5.900	24.800	3.600	3.450	3.450
13	6.025	5.750	24.500	3.600	3.450	3.450
14	6.025	5.600	24.750	3.600	3.450	3.450
15	6.025	5.550@5.650	24.500	3.600	3.450	3.400
16	6.025	5.550@5.650	24.500	3.600	3.450	3.400
17	6.025	5.550@5.650	24.500	3.600	3.450	3.400
19	6.025	5.550@5.650	23.875	3.600	3.450	3.400
20	6.025	5.500@5.525	24.375	3.500	3.350	3.250
21	6.025	5.500@5.525	24.625	3.400	3.250	3.250
22	6.025	5.750	24.750	3.400	3.250	3.250
23	6.025	5.750	24.800	3.400	3.250	3.250
24	6.025	5.750	24.850	3.400	3.250	3.250
26	6.025	5.750	24.725	3.300	3.150	3.250
27	6.025	5.700	24.450	3.150	3.000	3.225@3.250
28	6.025	5.650	24.750	3.000	2.850	3.250
29	6.025	5.650	24.600	3.00@3.05	2.85@2.90	3.200
30	6.025	5.675	24.500	3.00@3.10	2.90	3.150
Av. for Month	5.978	5.707	24.779	3.465	3.315	3.322
Sept.			**Average for Week**			
7	5.788	5.738	25.135	3.600	3.450	3.275
14	6.025	5.810	25.000	3.600	3.450	3.454
21	6.025	5.571	24.396	3.550	3.400	3.350
28	6.025	5.725	24.721	3.275	3.125	3.248
Sept.			**Calendar Week Averages**			
3	5.577	5.538	24.538	3.583	3.433	3.038
10	5.975	5.871	25.390	3.600	3.450	3.430
17	6.025	5.669	24.592	3.600	3.450	3.425
24	6.025	5.646	24.546	3.450	3.300	3.275

Silver, Gold, and Sterling Exchange New York and London

Sept.	Sterling Exchange: "Checks"	Sterling Exchange: "90-Day Demand"	Silver: New York	Silver: London	Gold: London
1	3.463750	3.468750	28.375	18.4375	118s11d
2	3.468750	3.472500	28.250	18.2500	118s 9d
3	3.472500	3.477500	28.500	18.4375	.....
5	Holiday			18.6250	118s 4d
6	3.475000	3.480000	28.875	18.8125	118s 6d
7	3.490000	3.495000	28.625	18.5625	118s 3d
8	3.485000	3.490000	28.375	18.3750	117s11d
9	3.485000	3.490000	28.125	18.0625	118s 2d
10	3.490000	3.495000	28.250	18.2500	.....
12	3.485000	3.490000	28.125	18.1250	118s
13	3.482500	3.487500	27.625	17.6250	118s 4d
14	3.481250	3.486250	28.125	17.9375	118s 3d
15	3.473750	3.477500	27.625	17.8750	118s 6d
16	3.471250	3.475000	27.625	17.7500	118s 7d
17	3.473750	3.477500	27.500	17.5625	.....
19	3.471250	3.475000	27.625	17.6875	118s 6d
20	3.470000	3.473750	27.500	17.6250	118s 8d
21	3.463125	3.467500	27.625	17.6875	118s10d
22	3.462500	3.466250	27.875	18.0625	119s 1d
23	3.465000	3.468750	27.500	17.7500	118s11d
24	3.462500	3.465625	27.375	17.6875	.....
26	3.453750	3.456250	27.500	17.8125	119s 1d
27	3.451250	3.454375	27.375	17.6875	119s 3d
28	3.451250	3.455000	27.500	17.7500	119s 5d
29	3.450000	3.453125	27.500	17.8125	.....
30	3.452500	3.455000	27.375	17.6875	.....
Av. for Month	347.003	.....	27.870	17.998	.....
Sept.			**Averages for Week**		
7	3.47400	.....	28.525	.....	.....
14	3.48479	.....	28.104	.....	.....
21	3.47052	.....	27.583	.....	.....
28	3.45771	.....	27.521	.....	.....

Calendar week averages: New York Silver, September 3rd, 28.542; 10th, 28.450; 17th, 27.771; 24th, 27.583.

London Market

eptember	Copper: Standard Spot	Copper: Standard 3 Mo.	Copper: Electrolytic (Bid)	Tin: Spot	Tin: 3 Mo.	Lead: Spot	Lead: 3 Mo.	Zinc: Spot	Zinc: 3 Mo.
1	35.2500	35.2500	37.50	147.750	149.000	12.8750	13.2500	15.0000	15.3125
2	35.6875	35.8125	38.00	153.750	155.000	13.3750	13.6875	15.3750	15.6875
5	37.3750	37.4375	39.00	154.750	156.250	13.8125	14.0625	15.9375	16.1250
6	37.8750	38.0000	40.00	157.250	158.750	14.0000	14.3750	16.2500	16.5000
7	37.6875	37.6875	40.00	157.625	159.250	14.3750	14.6250	16.5000	16.6875
8	38.0000	38.0625	40.00	157.750	159.250	14.0625	14.3750	16.3125	16.5625
9	35.7500	35.7500	39.00	155.750	157.250	13.1250	13.5625	15.5625	15.8125
12	36.0000	36.0000	39.00	155.000	156.750	13.4375	13.8125	15.6250	15.8750
13	31.5000	31.5000	35.00	147.500	149.000	12.3125	12.8125	15.0000	15.1875
14	33.3750	33.3125	37.00	151.625	153.250	12.7500	13.0625	14.8125	15.0625
15	33.2500	33.1250	37.00	149.500	151.250	12.5625	12.8750	14.7500	15.0000
16	34.0000	34.0000	37.75	151.625	153.250	12.9375	13.2500	15.2500	15.5000
19	32.6875	32.6875	37.25	147.125	148.875	12.6875	12.8750	15.0000	15.2500
20	32.8750	32.8750	37.25	149.500	151.250	12.6875	12.8750	15.2500	15.3750
21	34.7500	34.6875	38.00	151.750	153.500	12.9375	13.0625	15.3750	15.5625
22	35.6875	35.7500	39.25	154.375	156.125	13.3125	13.3750	15.7500	15.8125
23	35.1250	35.1250	39.00	153.750	155.500	12.8750	13.0000	15.5625	15.6250
26	35.3750	35.4375	39.00	153.125	154.875	13.1250	13.1875	15.5625	15.5625
27	34.2500	34.3750	38.50	151.500	152.625	12.6875	12.7500	15.0625	15.0625
28	34.6250	34.6875	38.50	153.500	155.000	12.9375	12.9375	15.3125	15.3125
29	34.5625	34.6250	38.50	153.250	154.750	12.8750	12.8750	15.2500	15.2500
30	34.0000	34.1250	38.50	151.750	153.250	12.9275	13.0000	15.5000	15.4375
Average for month	34.986	.....	38.318	152.705	.....	13.122	13.349	15.455	15.616

The United States quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices of domestic class are in cents per pound.

Copper, lead and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.35c. per pound above St. Louis, this being the freight differential. Contract prices for High-Grade zinc delivered in the East and Middle West are 1c. above St. Louis

Silver and Sterling Exchange

	New York		London Spot		Sterling Exchange	
	1931	1932	1931	1932	1931	1932
January	29.423	29.780	13.810	19.623	485.260	342.515
February	26.773	30.136	12.432	19.573	485.551	345.141
March	29.192	29.810	13.524	18.336	485.596	363.463
April	28.279	28.298	13.120	16.923	485.764	374.731
May	27.650	27.755	12.858	16.868	486.188	367.370
June	27.250	27.466	12.707	16.844	486.291	364.471
July	28.255	26.700	13.197	16.930	485.349	354.865
August	27.524	27.986	12.815	18.000	485.284	347.449
September	28.180	27.870	14.101	17.998	451.245	347.003
October	29.538	.....	17.153	.....	388.029	.....
November	32.223	.....	19.393	.....	371.130	.....
December	30.120	.....	20.023	.....	336.798	.....
Year	28.700	.....	14.594	.....	452.707	.....

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Copper *(a)*

	F.O.B. Refinery		London Spot			
	Electrolytic		Standard		Electrolytic	
	1931	1932	1931	1932	1931	1932
January	9.838	7.060	44.938	39.459	47.524	46.200
February	9.724	5.965	45.372	36.917	47.950	41.381
March	9.854	5.763	44.818	33.039	47.699	36.786
April	9.392	5.565	42.694	29.943	45.375	34.190
May	8.665	5.237	38.897	28.548	42.175	32.833
June	8.025	5.145	35.827	26.872	38.966	30.841
July	7.698	5.053	34.402	26.071	37.293	29.107
August	7.292	5.219	32.572	31.401	35.388	34.784
September	6.988	5.978	31.503	34.986	36.148	38.318
October	6.775	.....	34.957	.....	41.000	.....
November	6.558	.....	35.854	.....	41.190	.....
December	6.580	.....	38.273	.....	44.409	.....
Year	8.116	.....	38.342	.....	42.093	.....

New York quotations, cents per pound. London, pounds sterling per long ton.
(a) Export copper, f.o.b. refinery: Average for September, 5.707

Lead

	New York		St. Louis		London			
	1931	1932	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos.
January	4.802	3.750	4.604	3.550	13.872	13.905	15.084	15.128
February	4.552	3.712	4.340	3.499	13.444	13.550	14.560	14.571
March	4.527	3.150	4.276	2.993	13.128	13.355	12.345	12.634
April	4.412	3.000	4.164	2.900	12.375	12.606	11.223	11.503
May	3.818	3.000	3.651	2.900	11.491	11.778	10.673	11.036
June	3.917	2.993	3.761	2.896	11.582	11.952	9.608	9.898
July	4.400	2.747	4.225	2.611	12.731	12.899	9.818	10.152
August	4.400	3.235	4.225	3.093	11.944	11.994	11.349	11.588
September	4.400	3.465	4.217	3.315	11.932	12.026	13.122	13.349
October	3.964	.....	3.773	.....	13.227	13.270	.....	.....
November	3.937	.....	3.758	.....	14.577	14.491	.....	.....
December	3.792	.....	3.592	.....	15.188	15.361	.....	.....
Year	4.243	.....	4.049	.....	12.958	13.099	.....	.....

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Tin

	New York		London	
	1931	1932	1931	1932
	Straits		Spot	
January	26.137	21.804	115.798	140.219
February	26.315	22.018	117.919	139.143
March	27.065	21.863	121.852	129.810
April	25.222	19.244	112.775	108.935
May	23.221	20.948	104.331	122.286
June	23.478	19.659	104.966	114.530
July	24.978	20.931	111.478	125.863
August	25.738	22.985	114.875	142.017
September	24.618	24.779	117.813	152.705
October	22.723	.....	126.932	.....
November	22.779	.....	132.857	.....
December	21.328	.....	138.909	.....
Year	24.467	.....	118.375	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Zinc

	St. Louis		London			
	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos
January	4.035	3.011	12.747	13.113	14.416	14.834
February	4.012	2.817	12.303	12.694	13.872	14.289
March	4.002	2.787	12.190	12.676	12.616	13.024
April	3.717	2.725	11.353	11.838	11.670	11.958
May	3.306	2.532	10.484	10.875	12.432	12.682
June	3.416	2.777	11.270	11.750	11.548	11.866
July	3.893	2.537	12.280	12.802	11.592	11.967
August	3.817	2.758	11.444	12.028	13.594	13.844
September	3.744	3.322	11.571	12.063	15.455	15.616
October	3.377	.....	12.733	13.216	.....	.....
November	3.209	.....	13.845	14.247	.....	.....
December	3.149	.....	14.361	14.818	.....	.....
Year	3.640	.....	12.215	12.677	.....	.....

St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Cadmium and Aluminum

	Cadmium		Aluminum	
	1931	1932	1931	1932
January	67.115	55.000	23.300	23.300
February	55.000	55.000	23.300	23.300
March	55.000	55.000	23.300	23.300
April	55.000	55.000	23.300	23.300
May	55.000	55.000	23.300	23.300
June	55.000	55.000	23.300	23.300
July	55.000	55.000	23.300	23.300
August	55.000	55.000	23.300	23.300
September	55.000	55.000	23.300	23.300
October	55.000	.....	23.300	.....
November	55.000	.....	23.300	.....
December	55.000	.....	23.300	.....
Year	56.010	.....	23.300	.....

Aluminum in cents per pound, 99 per cent grade.
Cadmium, cents per pound.

Antimony, Quicksilver, and Platinum

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1931	1932	1931	1932	1931	1932
January	7.317	5.976	103.000	64.900	36.000	40.000
February	7.069	6.489	100.205	66.304	34.000	40.000
March	7.127	6.188	100.423	72.537	29.538	40.000
April	6.888	5.746	102.077	72.125	26.346	40.000
May	6.524	5.170	101.140	66.380	24.980	39.500
June	6.342	5.034	92.058	59.481	37.115	37.500
July	6.802	5.000	85.808	53.580	40.000	35.200
August	6.596	5.144	80.115	47.444	40.000	35.000
September	6.542	5.606	76.300	47.500	40.000	33.800
October	6.517	.....	72.385	.....	40.000	.....
November	6.679	.....	68.587	.....	40.000	.....
December	6.231	.....	66.115	.....	40.000	.....
Year	6.720	.....	87.351	.....	35.665	.....

(a) Antimony quotations in cents per pound, for ordinary brands. (b) Quicksilver in dollars per flask of 76 lb. (c) Platinum in dollars per ounce troy.

Pig Iron[1]

	Bessemer		Basic		No. 2 Foundry	
	1931	1932	1931	1932	1931	1932
January	17.50	16.00	17.00	15.00	17.00	15.50
February	17.29	15.68	16.79	14.68	16.79	15.18
March	17.00	15.50	16.50	14.50	16.50	15.00
April	17.00	15.50	16.50	14.50	16.50	15.00
May	17.00	15.50	16.38	14.50	16.44	15.00
June	17.00	14.85	15.50	14.35	16.00	14.75
July	17.00	14.56	15.50	14.00	16.00	14.50
August	17.00	14.50	15.50	14.00	16.00	14.50
September	17.00	14.50	15.50	14.00	16.00	14.50
October	16.92	.....	15.42	.....	16.00	.....
November	16.50	.....	15.00	.....	16.00	.....
December	16.25	.....	15.00	.....	15.75	.....
Year	16.96	.....	15.88	.....	16.25	.....

Iron in dollars per long ton. [1]F.o.b. Mahoning and Shenango Valley furnaces; freight to Pittsburgh, $1.76.

prices for the Prime Western grade.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

London prices for lead and zinc are the official prices for the morning session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

New York silver quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, 925 fine. Sterling prices represent forenoon market demand. Cables command a premium.

Current Statistics of Production and Stocks of Copper, Lead, and Zinc

Data from American Bureau of Metal Statistics, American Zinc Institute, and Metallgesellschaft.

All Figures Except Tin Represent Tons of 2,000 Lb.

	1931				1932							
COPPER — North and South America	Sept.	Oct.	Nov.	Dec.	Jan.	Feb.	March	April	May	June	July	Aug.
Production, refined, daily average	2,890	2,918	2,693	2,780	.....	.....	.....	.....	.....	.....	.....	.....
Domestic shipments	40,459	51,348	37,436	36,972	.....	.....	.....	.....	.....	.....	.....	.....
Foreign shipments	22,124	21,030	20,257	26,032	.....	.....	.....	.....	.....	.....	.....	.....
Stocks, blister and refined	658,321	.....	.....	722,014	.....	.....	.....	.....	.....	.....	.....	.....
LEAD — United States												
Production, refined, from domestic ore	31,966	36,546	31,671	33,576	32,180	28,081	30,345	23,236	25,902	26,068	15,819	17,118
Production, secondary and foreign	3,934	4,722	3,820	4,031	3,554	3,920	3,989	3,134	2,941	2,033	3,754	6,308
Production, total, daily rate	1,163	1,331	1,180	1,213	1,185	1,103	1,107	879	930	936	631	756
Shipments reported	34,059	34,276	31,216	30,297	27,867	26,319	31,162	26,081	24,258	21,511	19,723	28,248
Stocks, end of month	132,804	139,796	144,057	151,380	160,257	165,933	169,091	169,091	173,929	180,460	180,255	175,426
ZINC — United States												
Production, daily average	712	699	684	709	723	742	726	688	601	547	476	432
Domestic shipments	20,902	21,163	20,327	21,005	22,472	21,896	22,576	18,046	18,087	14 958	12,896	18,069
Export shipments	.....	.....	.....	.....	31	.....	.....	.....	.....	24	.....	39
Stocks, end of month	130,155	130,666	130,865	129,825	129,886	129,506	129,451	132,025	132,580	134,032	135,907	131,203
World Production Rate (Daily Average)												
Copper	4 023	3,940	4,070	3,940	.....	.....	.....	.....	.....	.....	.....	.....
Lead	3,654	3,806	3,980	3,976	3,938	3,872	3,598	3,587	3,711	3,497	3,223	3,361
Zinc	2,814	2,736	2,725	2,601	2,600	2,615	2,623	2,528	2,392	2,343	2,224	2,121
Tin (Metric Tons)	349	352	325	351	311	274	340	312	311	261	178	.....

Editor: A. W. ALLEN

Associate Editors: A. H. HUBBELL, H. D. KEISER

Assistant Editor: W. N. P. REED

Market Editor: H. H. WANDERS

Publishing Director: H. W. CLARKE

Engineering and Mining Journal

Contents for October, 1932

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McGRAW-HILL PUBLISHING COMPANY, INC.
330 West 42d St., New York, N. Y.
Cable Address: McGrawhill, N. Y.

McGRAW-HILL PUBLISHING COMPANY, LTD.
Aldwych House, Aldwych, London, W. C. 2, England
Cable Address: Backlash, Estrand, London

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